METHODS AND OPERATIONS AT THE YERINGTON COPPER MINE AND PLANT OF THE ANACONDA CO., WEED HEIGHTS, NEV.

By M. Clair Smith

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METHODS AND OPERATIONS AT THE YERINGTON COPPER MINE AND PLANT OF THE ANACONDA CO., WEED HEIGHTS, NEV.[1/]

by

M. Clair Smith[2/]

SUMMARY

This report is one of a series describing exploration, development, mining methods, and costs at United States open-pit mines; the Bureau of Mines prepares and publishes them in cooperation with the mining companies. Their purpose is to promote development and conservation of our mineral resources by disseminating technologic information on mining methods and practices at various United States mines.

The exploration, development, equipment, and mining of a large, oxidized-copper ore deposit by open-pit methods, the leaching of the ore with sulfuric acid solution, and the recovery of the copper as cement copper are described herein.

INTRODUCTION

The Yerington mine of The Anaconda Co. is at Weed Heights, Lyon County, Nev., at an elevation of 4,500 feet and is 80 miles by road southeast of Reno (fig. 1). Yerington, county seat of Lyon County, with a population of 1,500, is in the wide, level valley of the Walker River 2-1/2 miles east of Weed Heights. Wabuska, a station on the Hazen - Mina branch line of the Southern Pacific Railroad, is 12 miles north of Weed Heights.

The climate of the area is moderate, with an annual precipitation averaging 5 inches, occurring mostly as light rains; there is very little snow.

Weed Heights is a town comprising 250 new houses, 8 dormitories, a post office, and a recreation center; it was built by The Anaconda Co. to accommodate its employees.

The Anaconda Co. is mining a large, oxidized-copper deposit by open-pit methods at Weed Heights. The Yerington operation comprises stripping and mining an oxidized-copper deposit, leaching the ore with sulfuric acid, and recovering the copper by precipitation as cement copper, which is shipped to the company smelter at Anaconda, Mont., and then to Great Falls, Mont., for refining. Sulfuric acid is manufactured at Weed Heights from sulfur ore mined by The Anaconda Co. at the Leviathan mine in Alpine County, Calif., 58 miles west of Weed Heights. Figure 2 is a diagram of the Yerington operation.

1/ Work on manuscript completed May 1957.

2/ Mining engineer, Bureau of Mines, Region II, Reno, Nev.

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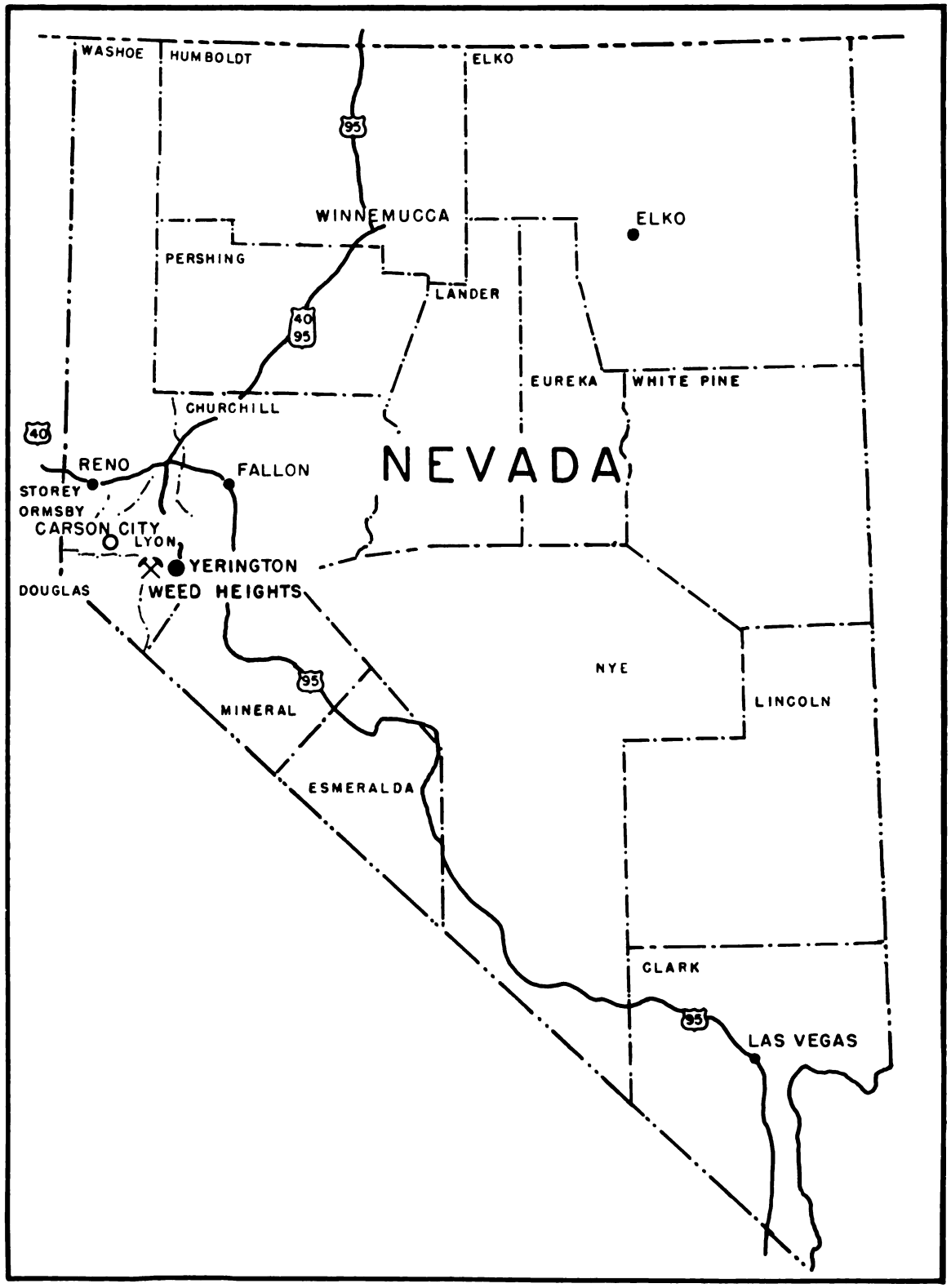

FIGURE 1. - Location Map, Yerington Mine, Weed Heights, Nev.

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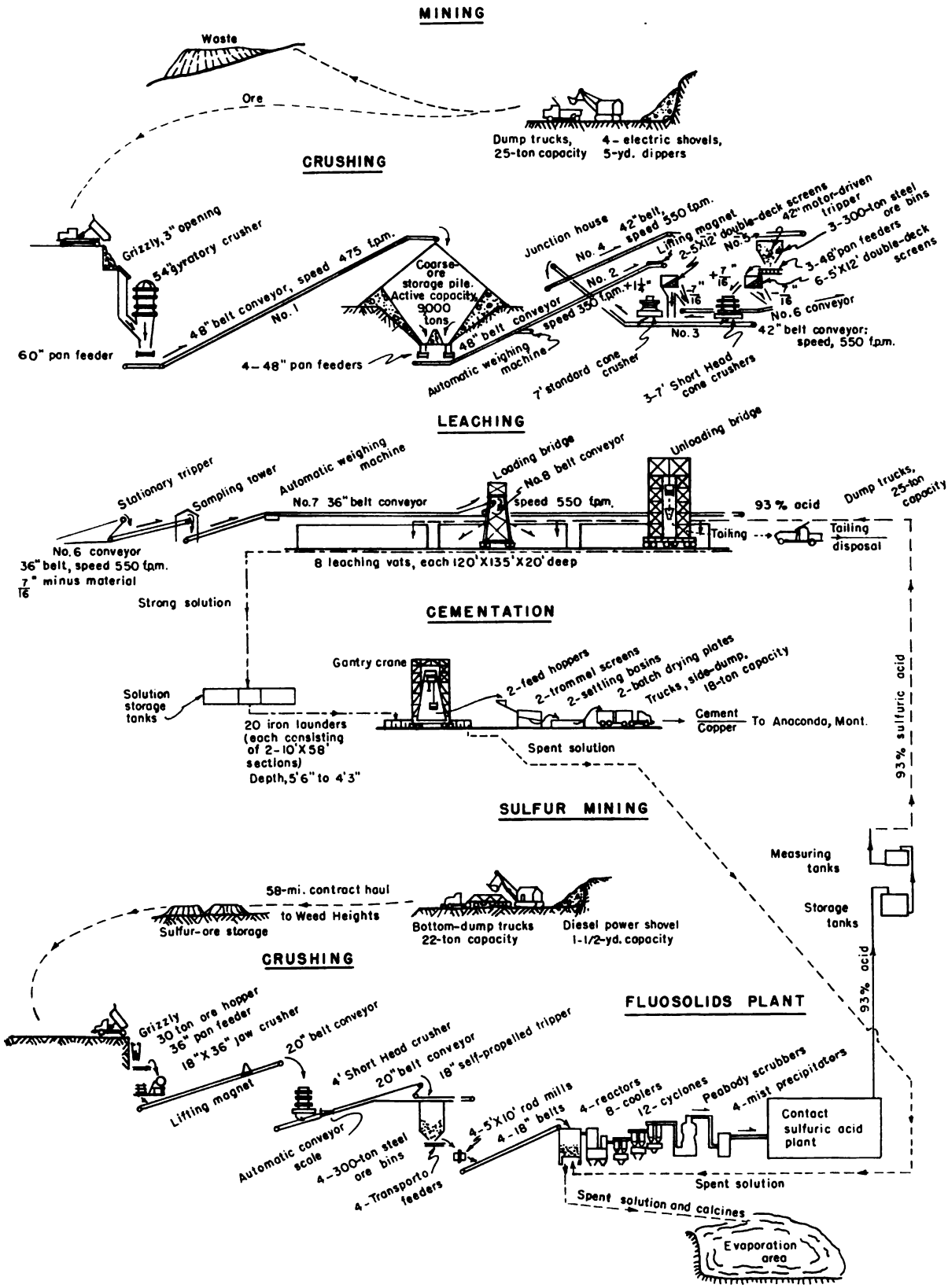

FIGURE 2. - Yerington-Mine Plant of The Anaconda Co.

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ACKNOWLEDGMENTS

Acknowledgment is made to The Anaconda Co. for permission to publish this paper. Special acknowledgment is due Albert E. Millar (general manager), H. R. Burch (assistant general manager), C. J. Houck (mine superintendent), A. J. Gould (plant superintendent), Don Gill (general mine foreman), L. B. McConville (chief office engineer), Mark B. Nesbitt (FluoSolids and acid-plant foreman), and K. W. Humphreys (personnel director) for their cooperation and assistance in compiling the data on the mining, leaching, and recovery of cement copper.

All photographs and maps were provided through the courtesy of The Anaconda Co.

HISTORY

The Yerington mining district, with Yerington at about the center of the district, extends east and west across Lyon County, north to include the deposits near Wabuska, and south to Wilson Canyon.

Mining in the district dates from 1865, when attempts were made to work the oxidized copper ore at the Ludwig mine. Between 1865 and 1907 intermittent operations within the district produced about 1 million pounds of copper.

Increased activity in the district began in 1907, and 100 million pounds of copper was produced to the end of 1932. In the early part of this period the Nevada Copper Belt Railroad was built from the mines to a smelter at Thompson and connected with a branch line of the Southern Pacific at Wabuska, Nev. A 1,800-ton-per-day smelter was constructed at Thompson, 2 miles north of Wabuska, by Mason Valley mines. This smelter was operated during 1912-14, 1917-18, and 1926-29. In 1926 a 1,000-ton concentrating mill was built, and a large output is recorded from the district from 1926 to 1929.

Mason Valley mines closed the smelter in 1929 and sold its mining interests. Production from the district was intermittent and small from 1929 to 1953. In November 1953 The Anaconda Co. began production.

The Yerington mine was previously known as the Empire-Nevada. A company organized in 1907 held 500 acres and at that time realized that the deposit was large and low grade, similar to the porphyry coppers of Arizona. In 1914 a limited drilling program by the Empire-Nevada Copper Mining Co. disclosed that the oxidized mineralization persisted to a depth of 500 feet, but the copper content was too low to be considered ore at that time. The company abandoned the property in the early 1930's. It was later acquired through tax sale and location by Messrs. Guild, Adams, and Herrin.

Anaconda took a lease and option on the property in 1941 and conducted an extensive exploration program from 1942 to 1945. Plans and cost estimates for mining the ore body were made.

At the outbreak of the Korean hostilities a contract was made between the Federal Government and The Anaconda Co., which required The Anaconda Co. to erect a complete plant, strip the ore body of overburden, and produce copper within 2 years. In turn, the Government allowed rapid amortization of 75 percent of the capital expenditures and guaranteed a price of not less than 25-1/2 cents per pound for the copper that Anaconda was unable to sell to industry, up to a maximum of 256 million pounds during the first 6 years of operations.

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The stripping of overburden was begun in June 1952, and the first copper was shipped in November 1953. The planned production of 5 million pounds of copper per month was attained in May 1954 and has continued at that rate. Figure 3 is a location map showing the various units of The Anaconda Co. operations. Figure 4 is an aerial view of the plant.

The reader is referred to other published articles related to The Anacona Co. Yerington operations by Ramsey,3/ Nordyke,4/ Mining World (July 1954, August 1956, and December 1956 issues),5/ and Burch.6/

DESCRIPTION OF DEPOSIT

The ore body is a disseminated porphyry-copper deposit. Copper mineralization is associated with an intrusion of quartz-monzonite porphyry into a granodiorite mass, the mineralization occurring in the quartz-monzonite porphyry and the bordering granodiorite. Secondary intrusions consist of andesite and rhyolite dikes. Part of the deposit has gravel overburden.

Chrysocolla is the chief copper mineral, with minor amounts of malachite, other copper silicates, and carbonates. The outlined ore within the deposit averages 0.82 percent copper. There is a narrow and vague secondary enrichment zone between the oxide and sulfide zones; in this respect, the deposit differs from most other copper deposits of the southwest in which the main ore bodies lie in the secondary enrichment zone.

The major differences that affect the mining of the Yerington ore body from other porphyry deposits are the widely varying copper content and its occurrence in smaller masses than is usual. Ore occurs in layers that dip to the north, conforming to the structural trend of the ore body. The ore layers, varying from 70 to 300 feet in thickness, are separated horizontally and cut vertically by waste, making selective mining necessary. The deposit strikes N. 63° W., and most of it is overlain with gravel varying from 0 to 200 feet in thickness (fig. 5). The ore body is very spotty, varying from the cutoff grade to as much as 6 to 8 percent copper, thus requiring the mining to be under close control of the engineering department.

PROSPECTING AND EXPLORATION

The property was explored from 1942 to 1945 under the direction of Alex M. McDonald. At present the exploration in this district is under the direction of James R. Wilson. Geologic mapping in the pit is carried on as mining progresses. Diamond drilling is done on contract, under supervision of the exploration department.

3/ Ramsey, R. H., Anaconda's Nevada Project - New Approach to Copper Mining: Eng. and Min. Jour., Vol. 155, Aug. 1954, pp. 71-92.

4/ Nordyke, Lewis, Mining With Anaconda in Nevada: Explosives Eng., November-December 1955; pp. 167-182.

5/ Mining World, Yerington-Anaconda's Latest Contribution to Copper Metallurgy: Vol. 16, July 1954, pp. 44-69. Anaconda Modifies Weed Heights Pit Blasting; Vol. 18, December 1956, pp. 54-55. How Fertilizer Cuts Anaconda's Blasting Cost at Weed Heights; Vol. 18, August 1956, pp. 56-57, and 88.

6/ Burch, H. R., Operating Problems at Yerington: Min. Cong. Jour., vol. 41, March 1955, pp. 62-64.

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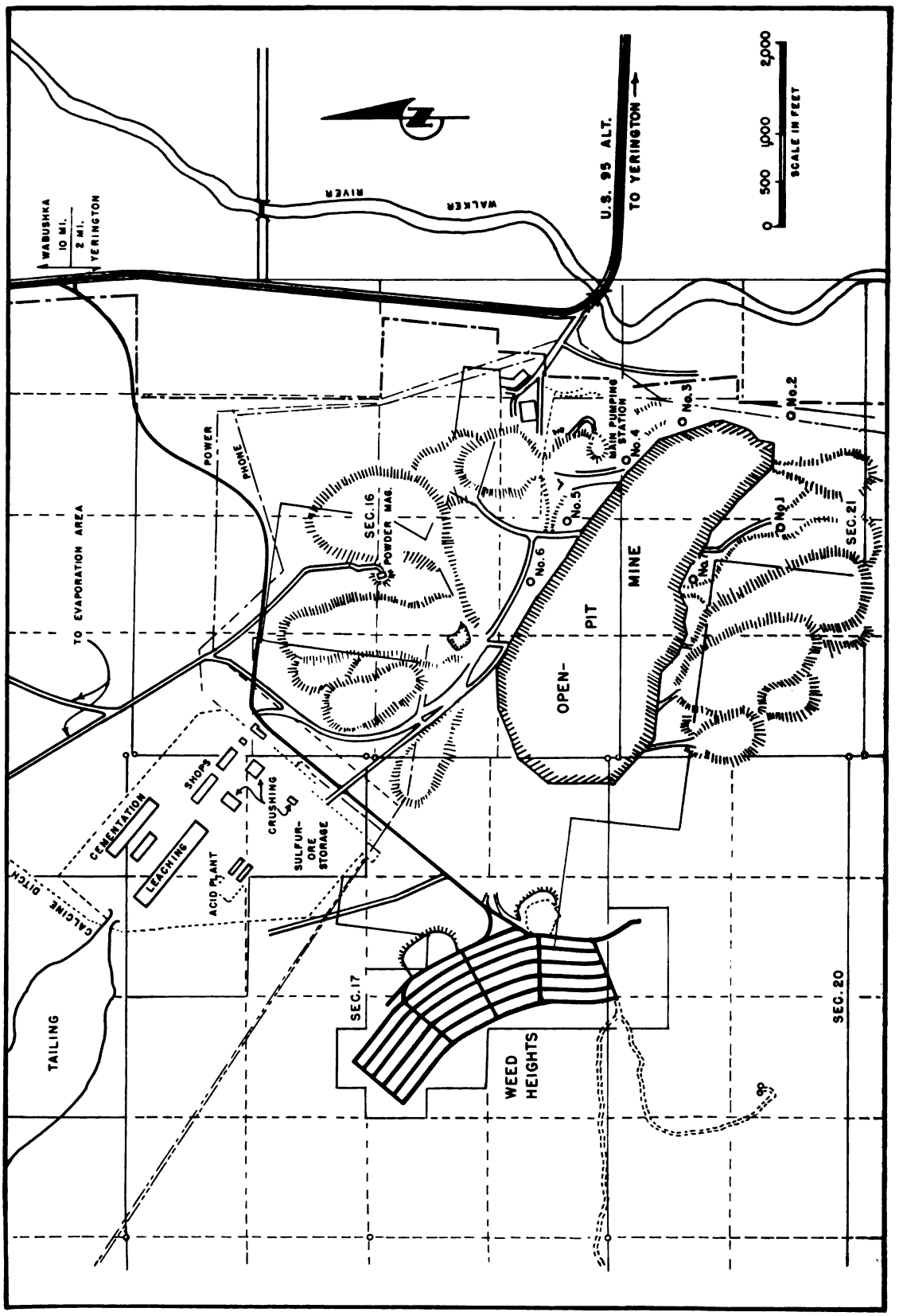

FIGURE 3. - Project of Anaconda Co.; Location of Various Units at Yerington.

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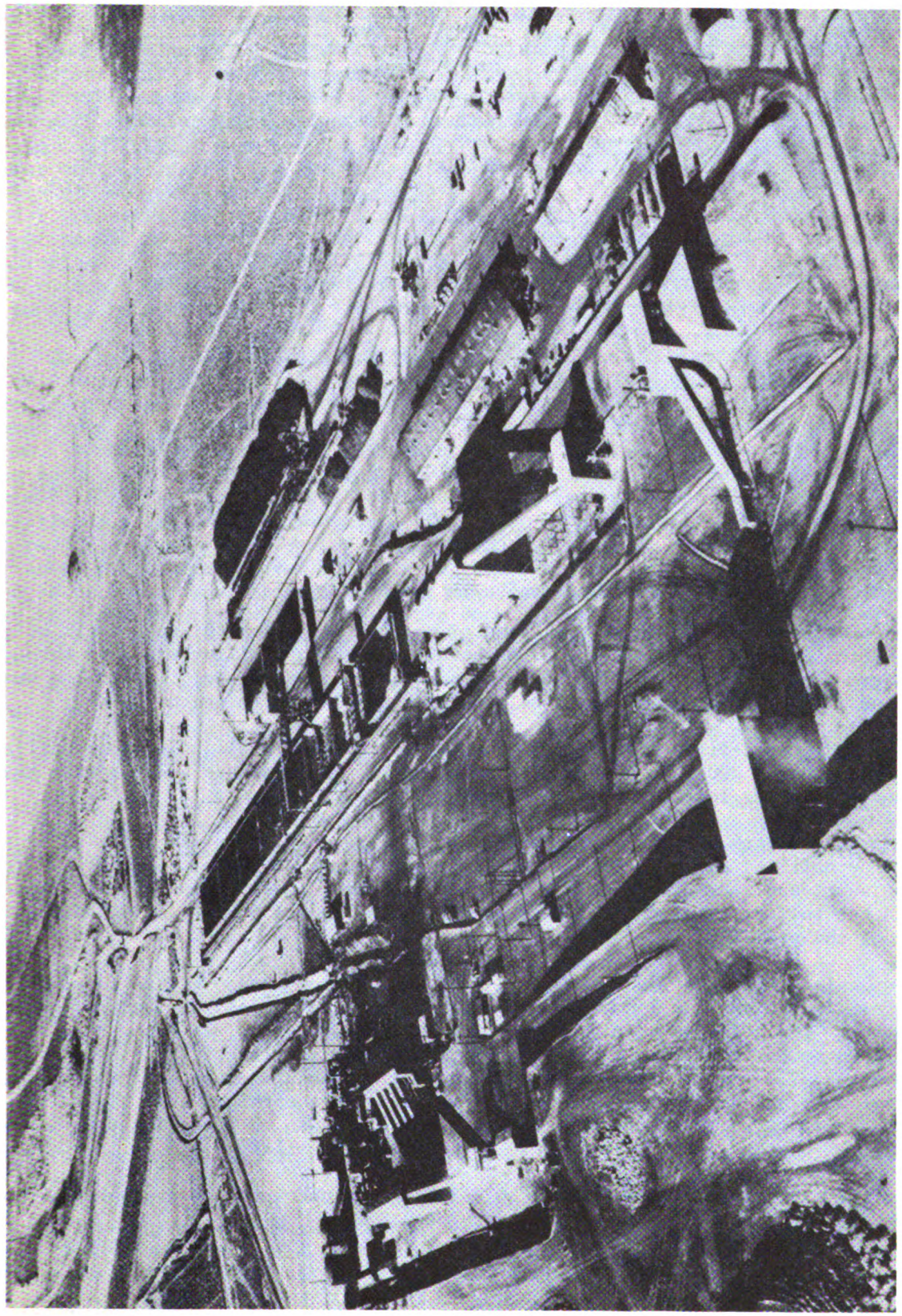

FIGURE 4. • Aerial View of The Anaconda Co. Plant.

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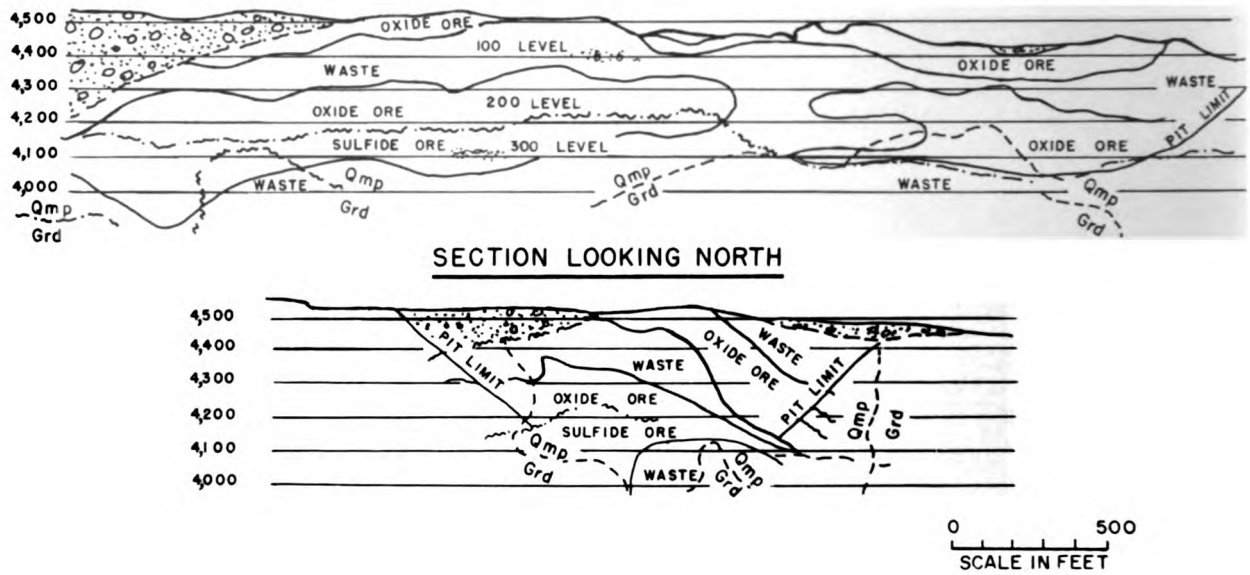

FIGURE 5. - Sections Showing Yerington Ore Body.

The preliminary exploration consisted of churn-drill holes, diamond-drill holes, a combination of both, and underground exploration. Churn drilling was done through the loose overburden into the solid formation to obtain a good seal. The holes were cased, and diamond drilling was used to obtain data relating to the ore body.

One-hundred-and-fifty diamond-drill holes and 10 churn-drill holes were drilled on the corners of a 200-foot-square grid pattern. Core and sludge samples were taken at 5-foot or shorter intervals.

Underground work was also done during the drilling period to check the returns obtained by drilling and to obtain samples for metallurgical tests. A shaft 400 feet deep was sunk, and 2 drifts, each 1,200 feet long, were driven, 1 in the oxide zone and 1 in the sulfide zone at elevations corresponding to the 4,275- and 4,125-foot benches of the pit (fig. 5).

Four raises totaling 350 linear feet in the oxide zone and 4 raises totaling 250 linear feet in the sulfide zone were driven from the 2 levels at the location of 8 drill holes. The total material mined in the drifts and raises was kept separate by rounds, each round being regarded as one sample. Assay results of the diamond-drill holes averaged 7.8 percent lower than the assays of the mined material from the raises in the sulfide zone and 3.6 percent lower than the assays obtained from the raises in the oxide zone. A representative portion of each round was obtained for assaying and metallurgical testing. The rejects were kept for future mill tests.

Initial explorations from 1942-45 by drilling and underground development were the basis on which the Yerington project was planned.

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PLANNING THE OPERATION

Upon completion of the exploration program in 1945, plans were made for mining the deposit. Plan maps were made showing the location of diamond-drill holes, underground exploration, coordinates, geology, assays, and all other pertinent data. Bench maps were made for each 50-foot difference in elevation, and 2 sets of longitudinal and cross sections were made at 200-foot intervals on a 100-foot scale. One set of these sections paralleled the exploration holes, the longitudinal sections being east-west and the cross sections north-south; the other set (the longitudinal sections,) paralleled the strike of the ore body, N. 63° W., and the cross sections at right angles to the strike. The Yerington pit was designed from data obtained by geological mapping and exploration data. The wealth of practical knowledge of open-pit mining acquired by Anaconda officials at other properties was a major asset in planning the operation.

The pit is 4,500 feet long, 1,800 feet wide, and 450 feet deep, with an overall final slope of 1.0:1.0. Benches are at 25-foot intervals and permanent haulage roads at 8-percent grade. The overall waste to ore ratio is 1.7:1.0 (fig. 6). It is estimated that the oxide ore body will last about 15 years at the present rate of production. Benches were initially planned at 50-foot intervals; but soon after stripping was begun, the design was changed to benches at 25-foot intervals. With 50-foot-interval benches, more material could be loaded from each shovel setup, but the shovels used can only dig and load effectively to a height of approximately 35 feet; therefore, the upper 15 feet of the bench must cave to be loaded. The 25-foot bench interval permits the shovel operator to obtain cleaner separation of ore and waste and better control of the grade of ore going to the plant. It also allows the operator to load from the full height of the bench, giving him control of the face slope, as the benches can be trimmed with the shovel dipper, thus promoting safety.

Another factor in favor of the 25-foot bench is: Blast holes, when drilled 10 feet back from the brow of a 25-foot bench with the face slopes at 0.7 to 1.0, have a burden of 27.5 feet at 25 feet in depth - about equal to the height, whereas the toe burden would be 45 feet with a 50-foot hole.

This increase in burden would necessitate drilling larger diameter blast holes or springing present-size holes to get the required amount of powder at the correct position in the hole. The 25-foot benches have resulted in better fragmentation of the rock, which also has reduced the mining cost.

CHURN DRILLING AND ESTIMATION OF GRADE

When stripping of the ore body was begun in June 1952, a second set of exploration holes was planned to gain additional information. As the gravel overburden was removed, churn-drill holes were drilled to the 4,300-foot elevation. These holes were spotted to form a 100-foot grid within the original 200-foot grid. To accomplish this pattern of drilling, the churn-drill holes were offset 50 feet in each direction from the original holes (fig. 7).

Churn drilling of the 100-foot grid to the 4,300-foot elevation was completed over the whole area and drilling to 4,200-foot elevation has been completed in the west part of the pit.

Drill-cuttings samples were taken every 5 feet. The weighted assay value for each 25-foot bench was calculated from the assays of the 5-foot samples. Each 100-foot square (fig. 7) has one of the initial diamond-drill holes at a corner (D.D.H.A.)

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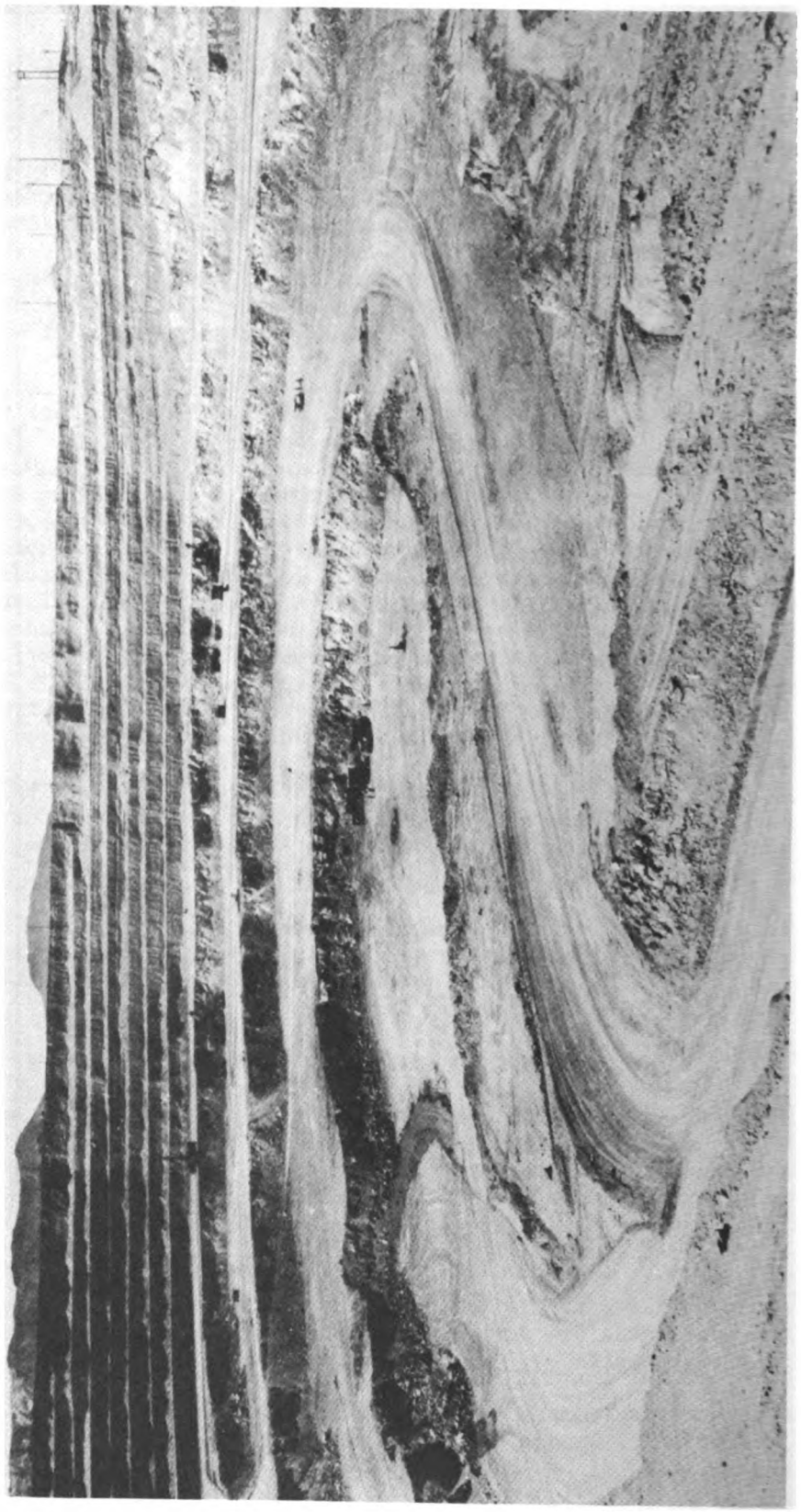

FIGURE 6. - View of Open Pit.

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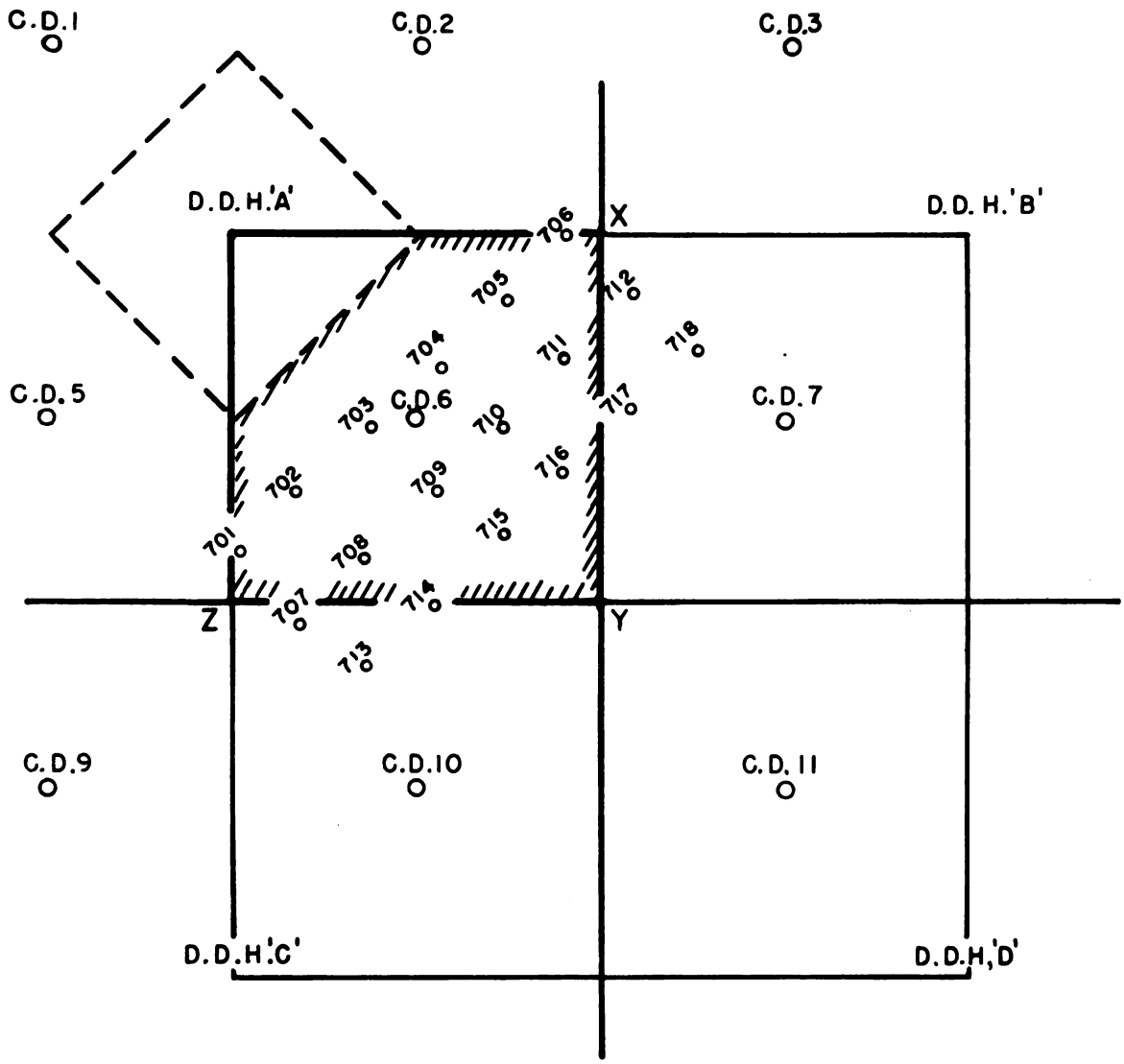

Weighted assay of block D.D.H.'A', X, Y, Z for preliminary planning equals 1/8 composite assay of D.D.H.'A' + 7/8 composite assay of C.D.6 for each 25' bench.

0 50 100
SCALE IN FEET

D.D.H–Initial exploration forming 200' grid (1942 – 1945)

O C.D.1–Churn-drill exploration to 4300' elevation forming 100' grid.

200' Grid

100' Grid

o 701–718 Rotary holes for blasting 20'–25' apart. Final estimate of ore grade is obtained from these holes.

FIGURE 7. - Exploration and Blasthole Drilling Patterns.

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and 1 of the churn-drill holes at its center (C.D. 6). In the plans, each 100-foot square (as D.D.H.'A', X, Y, Z) of a bench carries a weighted assay. The weighted assay is 1/8 of the composite assay of the 25-foot interval of diamond-drill hole (D.D.H.A.), plus 7/8 of the composite assay of the 25-foot interval of churn-drill hole C.D. 6 (fig. 7).

The weighted bench assays are used for planning; but as the benches are drilled for blasting the assays from the rotary blasthole samples are used in calculating the grade of ore going to the plant.

MINING

Planning

Rotary drills are used for drilling the solid material to be blasted. Each hole is numbered, sampled, surveyed, and plotted on the bench maps. Because these holes are only 20 to 25 feet apart and to the depth of the bench being mined, assays of these blast holes give closer control and are used in the daily mining plan.

The ore body is so variable in grade that mining must be carefully controlled, or impossible burdens will be placed on the leaching department. The mining, acid preparation, leaching, and cementation flowsheet must be run as a close-knit, coordinated unit to avoid serious difficulties.

Planning therefore must begin at the mine. The plans made by the engineering staff, after consultation with the mining department, are the center around which the operation revolves. Three sets of plans are maintained:

(1) The permanent overall mine plan, which shows actual accomplishment and planned pit boundaries at completion of mining.

(2) A set of plans detailing operations for the next 2 months.

(3) A plan predicting operations 6 months in advance.

All these plans show the pertinent data and undergo constant revision.

The important daily planning of the engineering department is the scheduling of the next day's mining. This schedule shows the amount and grade of material to be removed from the benches at specified places. It also shows where each shovel will work, the amount and grade of material it is expected to move, and the destination of the material. The shovels are operated 2 shifts, 6 days a week, removing approximately 220,000 tons of material.

Mine Equipment

Equipment purchased for the Yerington mining operation is as follows:

4 5-yard P. H., Model-1500 electric shovels.
1 1-1/2 P. H., Model-650 diesel shovel.
2 Joy rotary drills, Model-58 B. H.
30 25-ton, end-dump trucks, Dart Model-140, 300-hp. motors.
3 portable compressors - 1-600 c.f.m.; 2-315 c.f.m.
2 Bucyrus-Erie, Model 42T electric churn drills.
6 D-8 Caterpillar tractors with bulldozer.

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1 Super C Turnadozer.
1 Caterpillar 80 scraper.
2 Diesel-powered water trucks.
2 Caterpillar road patrols.
1 lubrication truck.
1 winch truck.
1 Tractor-trailer fuel truck.
2 1-ton utility trucks.
3 1/2-ton pickup trucks, and other miscellaneous equipment.

Some of this equipment is shown in figure 8.

Mine Control

Permanent triangulation stations were established around the pit, with secondary control stations within the pit. A new station may be established from either the permanent or secondary control systems.

The pit is designed with 2 permanent roadways, 40 feet wide; 1 is along the south side leading to the gravel overburden piles, the other along the north side leading to the crushing plant and the submarginal ore piles. Two diesel-powered water trucks and 2 Caterpillar road patrols maintain the haulage roads in excellent condition. A plan and cross section of the permanent roads from one bench to another are sketched in figure 9.

The design and engineering control of the benches to excavate them to permanent slope are shown in figure 10. It should be noted that the faces of the benches are dug about 5 feet beyond the permanent slope line and that the brow extends into the pit 10 feet past the permanent slope line, leaving a berm 15 feet wide at each bench level. The management believes that this makes a more stable and safe pit slope.

A section showing engineering control for temporary roads is shown in figure 10.

Daily Ore Control

The management determines the grade of ore to be treated at the plant. The engineering department is responsible for obtaining the grade desired.

Holes for blasting the benches are drilled from the top of each bench to a depth of 30 feet and spaced 20 to 25 feet apart in each direction, well in advance of the mining. A sample is taken from each hole and assayed; from the assays of these blast holes, the daily tonnage schedule is prepared.

Each afternoon the pit foreman receives from the engineering department sketch maps of portions of the pit, designating where the next 2 shifts of mining are to be done. These sketches give the grade of material, location, and approximate tonnage of each grade to be mined. A worksheet is also made out, suggesting the amount to be mined by each shovel, the grade of ore it will load, and the number of trucks to be assigned to each.

The engineering department plans for hauling to the crushing plant about 14,000 tons of ore that will average about 0.82 percent copper. A surge pile in excess of 12,000 tons must be built up during the week to supply ore for a leaching vat on Sunday. To accomplish this at the Yerington deposit (which is very variable in grade) and to keep the shovels mining at the desired benches without too many or too

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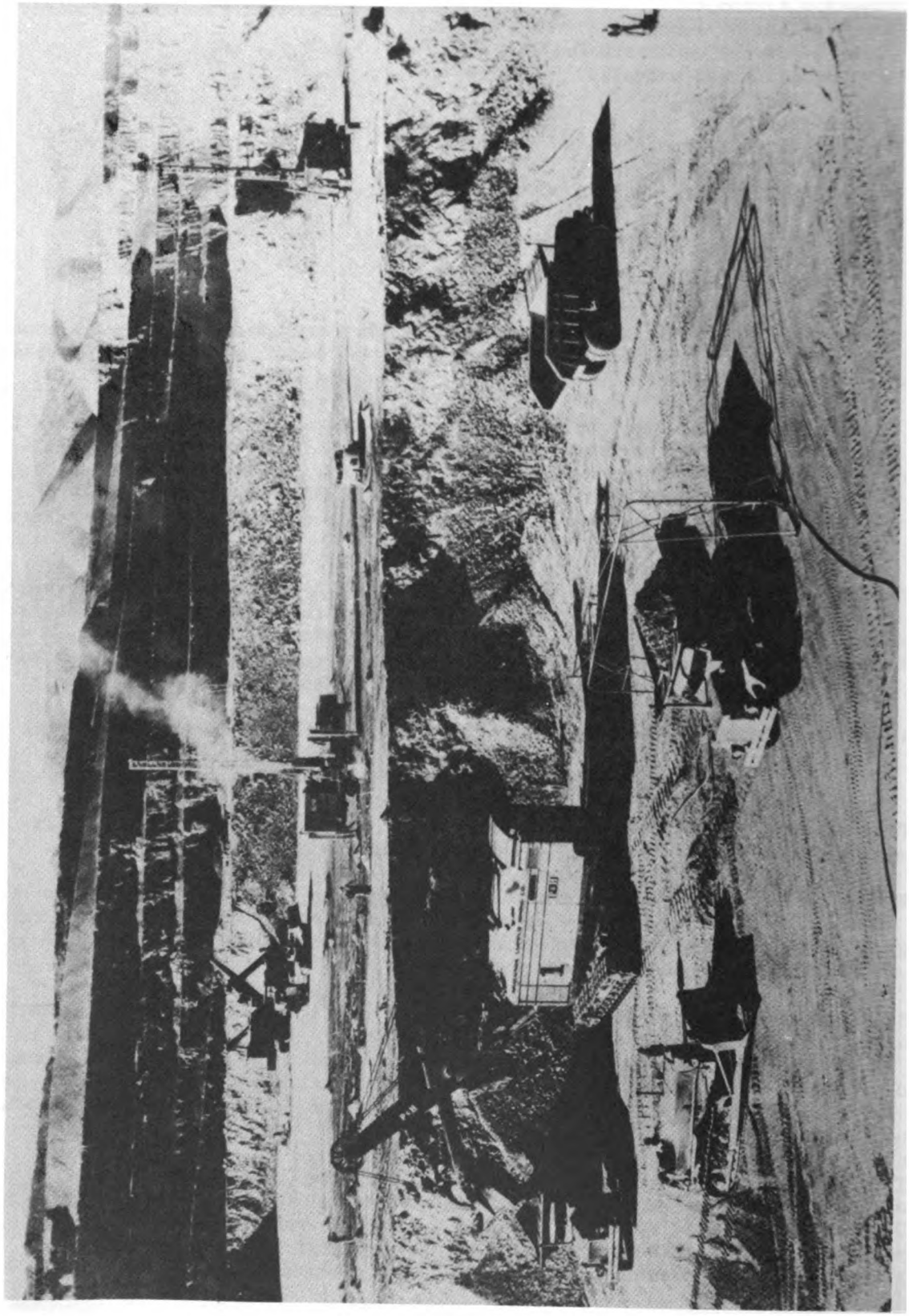

FIGURE 8. - Pit and Mining Equipment.

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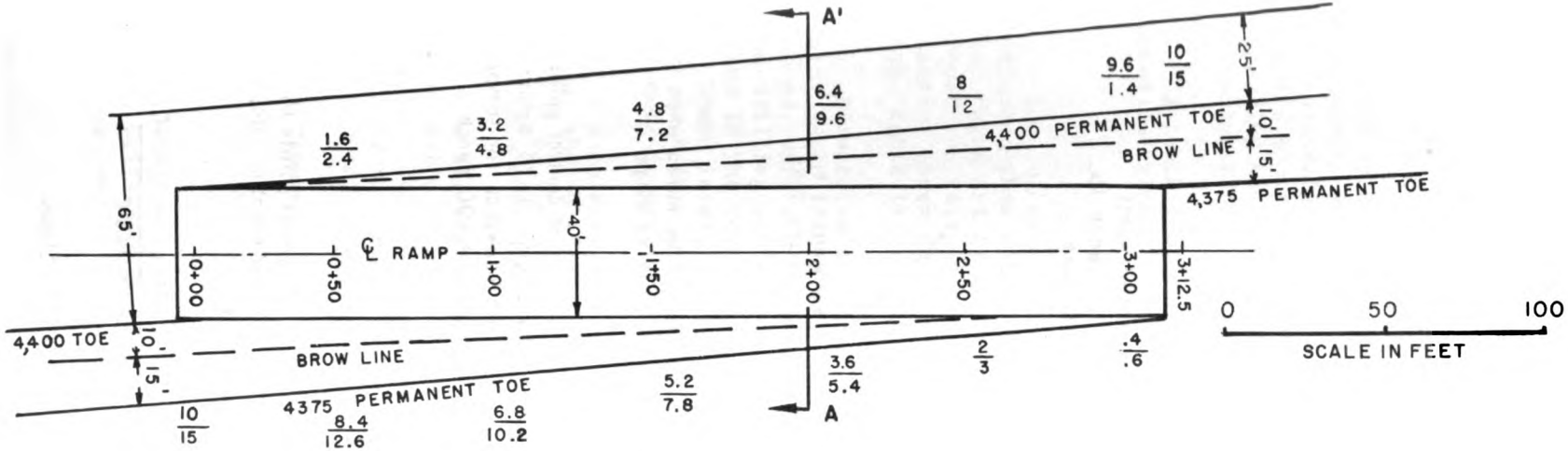
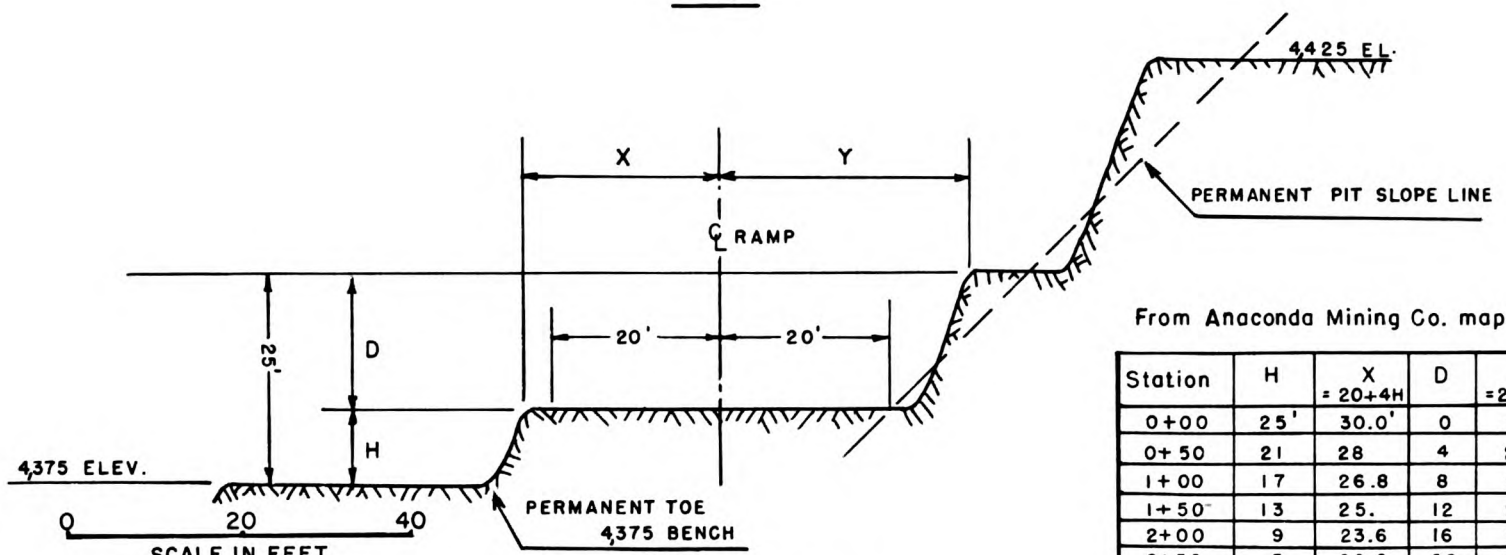


PLAN



SECTION A — A'

From Anaconda Mining Co. map

Station	H	X = 20+4H	D	Y = 20+6D
0+00	25'	30.0'	0	20.0
0+50	21	28	4	22.4
1+00	17	26.8	8	24.8
1+50	13	25.	12	27.2
2+00	9	23.6	16	29.6
2+50	5	22.0	20	32.0
3+00	1	20.4	24	34.4
3+12.5	0	20.0	25	35.0

FIGURE 9. - Plan and Cross Section of Permanent Roadway.

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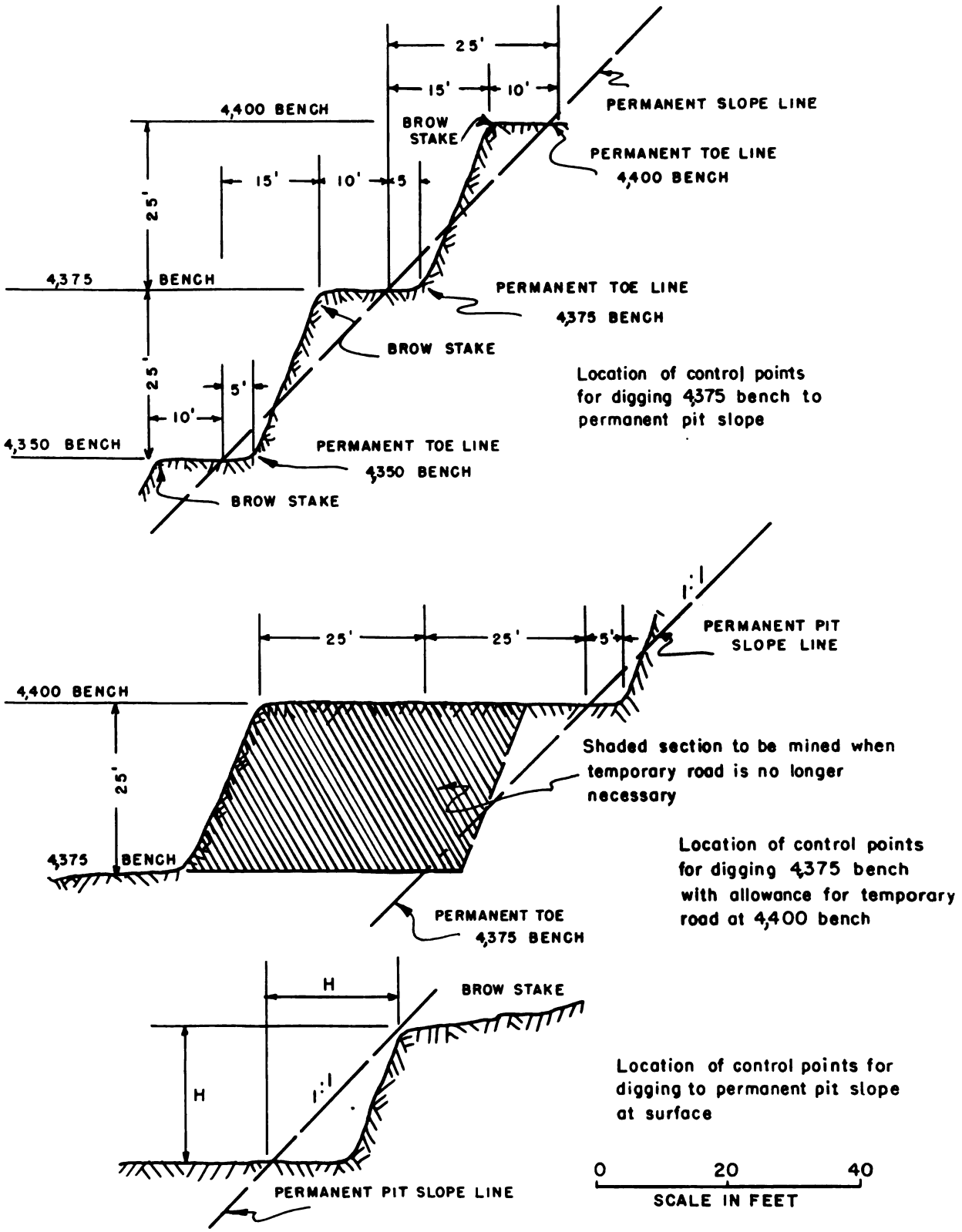

FIGURE 10. - Engineering Control for Excavation to Permanent Pit Slope.

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long a move requires detailed planning. A variation of 0.1 percent copper in ore mined and the grade estimated for the day is common. A variation of 0.02- to 0.03-percent copper content in the plan and actual results is usual for a month's operation.

Planning of each day's work considers the amount, grade, and kind of material to be moved and the operating factors.

The operating factors that have been gained from experience and are used by the engineering and mining department in planning daily operations are: A 5-yard shovel that excavates about 7,000 tons of material per shift; a 1-1/2 yard shovel that excavates about 2,000 tons per shift; and trucks that haul 650 to 1,700 tons per shift, depending upon the location in the pit where they are loaded, and the length of haul. Normally, 6 to 8 trucks are assigned to each shovel, but it may vary from 4 to 11.

A truck factor obtained by experience, time, and performance study for different locations within the pit aids in dispatching the number of trucks to a shovel.

Drilling and Blasting

Soon after mining was begun, the fractured condition and softness of the rock indicated that rotary drills with roller tricone bits could be used more efficiently than churn drills for blasthole drilling. Trial drilling with the rotary drills proved the rate of penetration to be 5.6 to 1 in favor of the rotary drills; consequently, churn drilling was completely replaced by rotary drilling (see statistical data, table 1).

Blastholes are drilled with track-mounted, Joy B. H. Champion rotary drills, using 7-3/8-inch bits. The holes are spotted by the engineering department and spaced, with regard to the character and formation of the ground, but ordinarily the drilling pattern consists of 3 rows of holes 20 to 22 feet apart, with the holes in the row 25 feet apart. The holes are drilled to a depth of 5 feet below the bench level, or approximately 30 feet. The first row of holes is collared 10 feet back from the brow of the bench. An average of 300 feet of hole per shift was made in drilling 617,716 feet to July 1, 1956.

A sample weighing 20 to 25 pounds is taken from the drill cuttings of each hole. A tool shaped like a pie server is pushed into one side of the cuttings and removes a segment of the drill cuttings. These cuttings are dry; are removed by air, and collect in a cone-shaped pile around the hole.

The blastholes are numbered, surveyed, and plotted on the bench maps. The holes are numbered in the order drilled from 1 to 10,000, then a new series is started with 1 again. Ordinarily, 3,000 feet or 100 holes are available for blasting. The holes available for breaking ground are spotted on the map with an open circle. An average drilling and blasting pattern is shown in figure 11.

A "powder" crew of 1 foreman and 4 men on day shift do all the blasting and any necessary secondary drilling. The location and the numbers of the holes to be blasted are designated by the pit foreman. Ordinarily, 3 rows of holes consisting of 10 to 50 holes, but averaging 30 holes, are blasted at one time.

In 1956 blasting experiments were made using different combinations of Fertilizer-grade, prilled ammonium nitrate, oxidizing agents, manufactured ammonium nitrate explosives, and 60-percent special gelatin dynamite as the detonating agent.

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TABLE 1. - Open-pit mine operations, June 1956 and total accumulated to July 1, 1956

	June 1956		Total accumulated to July 1, 1956	
Ore to plant:				
From benches	1/380,952	(0.80%)	10,448,666	(0.87%)
From stockpile			594,675	(.83)
Total tonnage	380,952	(.80%)	11,043,341	(.87%)
Ore stockpiled:				
Ore to stockpile			1,899,656	(.78%)
Stockpile to plant			594,675	(.83)
Total remaining			1,304,981	(.75%)
Low-grade material:				
0.40 - 0.60			4,564,128	(.49%)
.20 - .40	288,925	(0.28%)	13,015,005	(.31)
Total	288,925		17,579,133	
Waste mined:				
0.00 - 0.20	31,525	(.12%)	1,004,942	(.71%)
.00 ton gravel	352,625		17,593,582	
Total tonnage	384,150		18,598,524	
Total tonnage of material removed from pit:	1,054,027		48,525,979	
Shovels, electric:				
Total tonnage loaded	1,008,692		48,542,176	
Operating shifts	141.125		6,794.250	
Tons per operating shift	7,148		7,145	
Shovels, diesel:				
Total tonnage loaded	45,335		443,013	
Haulage trucks (large series):				
Total tonnage loaded	1,054,027		48,843,817	
Operating shifts	1,107		50,464	
Tons per operating shift	952		968	
Haulage trucks (small series):				
Total tons loaded			141,372	
Scrapers:				
Total tons loaded			135,465	

	June 1956	Total accumulated to July 1, 1956
Rotary drilling:		
Blasthole drilling:		
Total footage drilled	15,000	617,716
Footage lost	385	11,872
Feet per hour operated	52.77	52.35
Feet per operating shift	324.32	299.92
Operating shifts	46.250	2,059.595
Prospect drilling:		
Total footage drilled	1,415	154,507
Feet per operating shift	246.09	263.84
Operating shifts	5.750	585.605
Churn drilling:		
Blasting:		
Total footage drilled		16,136
Footage lost		1,377
Footage blasted		14,759
Feet per operating shift		53.5
Operating shifts		301.5
Prospecting drilling:		
Total footage drilled		77,356
Feet per operating shift		39.59
Operating shifts		1,953.83
Blasting:		
Tons broken by explosives	701,402	30,393,072
Pounds of powder used	77,427.50	3,818,167.50
Tons per pound of powder	9.06	7.96
Footage blasted	14,860	
Footage avail. beginning mo.	2,940	
Footage avail. during mo.	14,615	
Footage on hand end mo.	2,695	
Water wells:		
Footage drilled	275	4,683
Footage lost		650
Footage available	275	4,033
Cu. ft. water pumped	15,094,000	
Cu. ft. water consumed	10,682,000	
Cu. ft. water to Walker River	4,412,000	

1/ Short tons. Copper content shown as percent.

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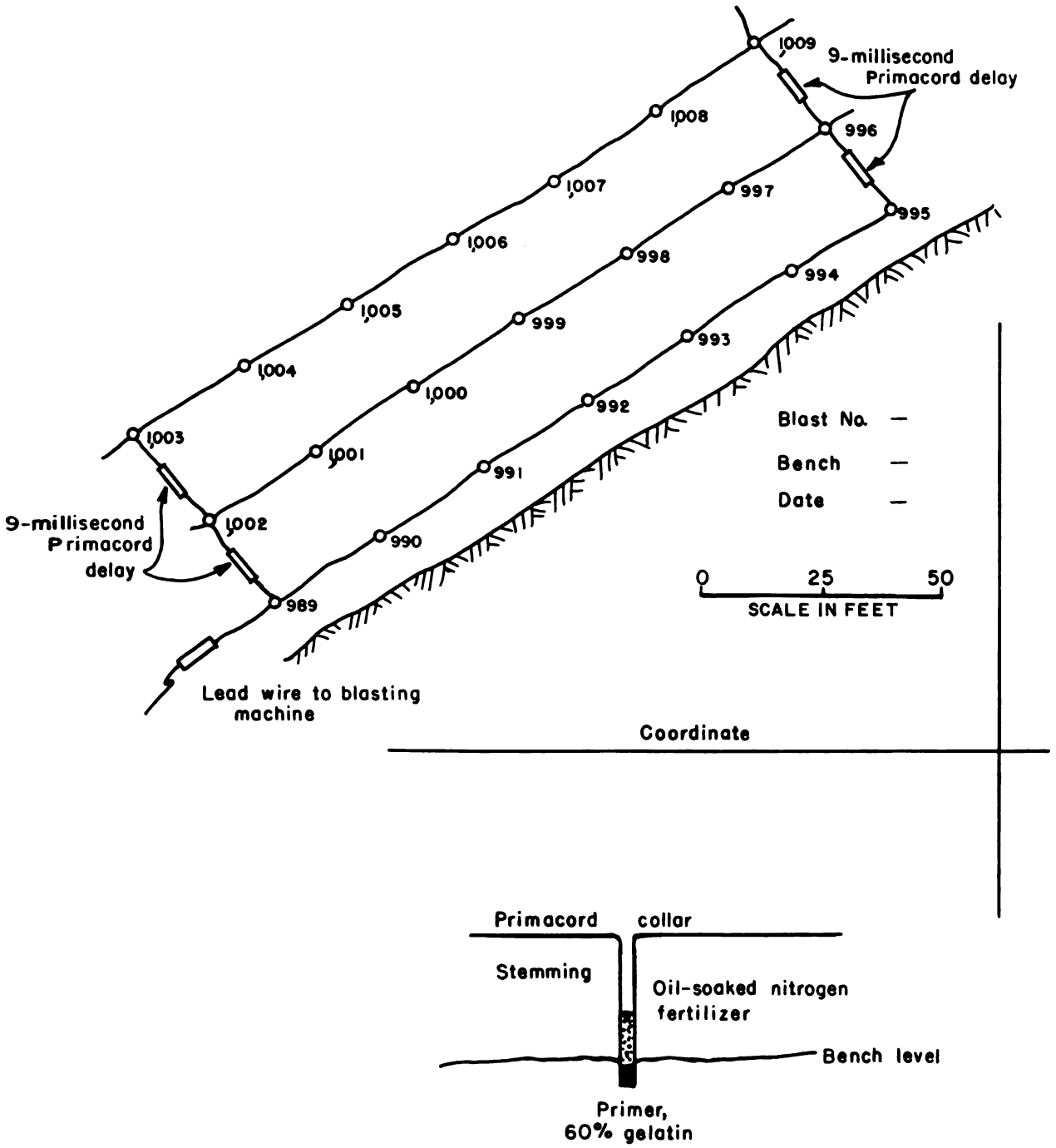

FIGURE 11. - Drilling and Blasting Pattern.

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After a year's experimentation it was proved that the best blasting results were obtained by using Fertilizer-grade, prilled ammonium nitrate with diesel oil as the oxidizing agent, with a small amount of special gelatin as the detonating agent. The costs of blasting with this combination of materials are approximately 50 percent of the former cost when manufactured ammonium nitrate explosive was used.

The explosives are distributed by truck near the holes to be blasted; each hole of average depth and burden requires 12-1/2 pounds of 60-percent gelatin dynamite and 2 sacks (160 pounds) of Fertilizer-grade ammonium nitrate. The sacks of ammonium nitrate are split open, and 1 gallon of diesel oil is added. The holes are loaded and connected with Primacord. Each row of holes is separated from the other by millisecond delays. The complete round is detonated with one electric cap at the end of the trunkline of Primacord leading to the holes in the first row (fig. 11).

The electric cap is wire-connected to an electric blasting box a safe distance from the blast. Ordinarily, the ground is well fractured by the blast, with very little flying rock.

The advantages of placing the primer in the lower part of the hole are that the stronger part of the explosive charge is at the heaviest burden point and that such placement is safer. If a missed hole occurs, the sensitive part of the charge is below grade, and only Fertilizer-grade ammonium nitrate remains in the broken rock to be handled by the shovel.

The explosives ratio is figured at 1 pound of explosive for 5 tons of burden, as the ground is drilled; but upon mining, the ratio is nearly 1 pound of explosive to 7-1/2 tons of material removed. The increase is due to backbreak of the holes and the digging of some material that does not need blasting in the pit.

Loading and Hauling

Five power shovels and thirty 25-ton trucks are available for loading and hauling. This equipment includes four 5-yard, P. H.-1500 electric shovels, one 1-1/2 yard, P. H.-655 diesel shovel, and thirty 25-ton, Dart end-dump trucks (fig. 12). Three 5-yard shovels and necessary trucks on day shift, two 5-yard shovels, one 1-1/2-yard shovel, and necessary trucks on swing (afternoon) shift are required to mine and haul the necessary tonnage.

Each shovel is operated by an operator and an oiler. One man is also employed at each shovel spotting the empty trucks, thus minimizing lost shovel time. One D-8 Caterpillar-tractor bulldozer with operator is kept near each shovel, cleaning up the scattered rock and pushing it against the broken face, making it easier and quicker for the shovel operator to obtain a full dipper of rock. Where the material being moved by a shovel is either ore or waste, the shovel operator directs the truck driver where the load is to be taken by a signal.

The gravel overburden is hauled out of the ore area to the south of the pit where it is piled. The rock stripped from the ore body is divided into 3 categories and piled separately; the material containing less than 0.2 percent copper is classed as waste and piled farthest from the plant to the north of the pit; rock containing 0.2 to 0.4 percent copper is piled separately nearer the plant, and material containing 0.4 to 0.6 percent copper was previously stockpiled near the plant. At present, it is mined as ore and goes directly to the plant. Some future day these low-grade stockpiles may be treated to recover the copper content.

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FIGURE 12. - Shovel Loading Trucks.

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A grab sample is taken from each load going to the submarginal piles and deposited in a composite sample box for that pile. At the end of each shift the material in each box is well mixed and a sample taken for assaying. A record is kept of the amount and assay of each stockpile. Table 1 shows the amount of the different grades of material removed from the pit in June 1956, the accumulated total tonnage to July 1, 1956, and the total performance of the equipment used in mining the deposit.

RADIO COMMUNICATION

Short-wave two-way radio communication is now considered a necessary installation in open-pit mining operations.

A 500-watt system with a 90-foot control tower is maintained at Weed Heights. Three dispatch units for governing and regulating calls are located at the garage office, machine-shop office, and electricians' office.

Communication units are installed in 15 mobile units (cars, trucks, etc.) and at the desks of the general manager, mine superintendent, chief engineer, and plant superintendent.

This method of communication between supervisors is convenient and rapid, saving many trips to coordinate the work.

Changing of routine operating procedure, calling maintenance and repair crews, transmitting work orders, reporting accidents and giving blast warnings are easily and quickly handled by the two-way radio system.

WATER CONTROL

While initial exploration was in progress, it was determined that the water table at the ore deposit was 100 to 125 feet below the surface at approximately the 4,380-foot elevation, the same as that of the Walker River, flowing northerly within 1,400 feet of the eastern limits of the pit. The water table within the pit area is not affected by the river. As the pit was designed to go below the water-table elevation, some means of controlling the water or lowering the water table was necessary.

Three wells (fig. 3, p. 6) were drilled by contractors at the eastern end of the pit, and pumps were placed in them.

The rate these wells were pumped, the elevation of the water level in the shaft within the pit, and the lowering of the water table in each well were recorded in water elevation records. It was decided that pumping the wells would control the water level. Four additional wells were drilled by The Anaconda Co. at the perimeter of the pit and farther west. Samples were taken and logs were made of the material penetrated by the company-drilled wells.

At present, water is being pumped to a reservoir from 3 of the wells, each at a rate of 700 to 900 gallons per minute. The water is then pumped to the housing project and mill. By pumping at the above rate it supplies the housing project and the mill with enough water and holds the water table 10 feet below the lowest point in the pit, which, at present, is the 4,250-foot bench. Since the start of operation, the water table within the pit area has been lowered from the 4,380- to 4,240-foot elevation, or 140 feet.

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As the water table is lowered, the wells are deepened and completed at 14-inch diameter to allow pumps to be placed in them. No serious water problems are anticipated.

EXPLOSIVES STORAGE MAGAZINE

An explosives storage magazine is built on a concrete slab at a safe but convenient distance from the pit and plant. The floor, ceiling, and exterior and interior walls are finished with tongue-and-groove boards. The space between the inner and outer walls is filled with weak concrete mortar. The outside walls and roof are covered with corrugated iron. Three doors at the front of the building add convenience to the unloading and storing of the explosives in an arrangement that leads to using the older explosives first.

The explosives magazine, between 2 pit dumps, has dirt banks on 3 sides to the height of the building. In case of an explosion, most of the explosive force would be directed upward, causing little damage.

A detonator storage building constructed of heavy timbers covered with earth and equipped with a heavy steel door is located 500 feet from the explosives storage magazine.

POWER

Electric power is furnished by the Sierra Pacific Power Co., which built a new powerline approximately 70 miles long from its transmission line near Washoe, Nev., to Weed Heights, aided by The Anaconda Co.

The power is reduced from transmission voltage through a 7,500-kv.-a. transformer bank and distributed to the plant and mine areas at 2,300 volts. It is further reduced in voltage to that required for the equipment served.

The average power requirements at the plant, mine, and townsite amount to 3,000,000 kw.-hr. per month.

EMPLOYMENT AND WAGE RATES

Normally, 450 daily-wage employees and 72 staff members are employed. Working agreements have been consummated between the unions and The Anaconda Co., and all wages of days'-pay employees are on a daily basis.

The daily wage schedules in effect September 1, 1956, are shown in table 2.

SAFETY PRACTICES AND FIRST-AID TRAINING

A safety committee composed of employees chosen from the different operational divisions inspects a portion of the plant, mine, and townsite every 2 weeks. Any unsafe practice or condition is noted, and recommendations are made to the management.

Safety suggestion boxes are installed in suitable places, and suggestions from all employees to eliminate any dangerous practice or condition are solicited. These suggestions are reviewed by the safety committee; and the suggestion, with the committee's recommendation, is given to the management.

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TABLE 2. - Daily wage schedules, 8-hour-day basis, in effect September 1, 1956, at the Yerington mine of The Anaconda Co.

Job classification	Daily rate	Job classification	Daily rate
Acid plant and FluoSolids:		Janitor	$15.27
FluoSolids plant helper	$16.07	Joy rotary driller (58 BH)	18.48
Acid-plant helper	16.52	Joy-rotary-driller helper	16.52
Acid-plant repairman	17.01	Laboratory assistant	15.62
Rod-mill operator	17.50	Laborer	15.27
FluoSolids-plant operator	17.99	Leadburner	20.44
Acid-plant operator	18.48	Lubrication, greaseman	16.07
Assayer	17.50	Machinist	18.97
Blade operator	17.99	Pitman	15.62
Bulldozer operator:		Powderman	17.50
1st class	18.97	Pumpman	17.01
2d class	17.99	Radio and maintenance B	17.50
3d class	17.50	Rigger-truck crane	17.50
Churn-drill operator	17.99	Sampler and sample bucker	16.07
Churn-drill helper	16.07	Shovel oiler:	
Crane operator	18.97	Class A	17.99
Crusher-conveyor lubricator	16.29	Class B[1/]	17.50
Crusher operator	17.50	Shovel operator	20.93
Crusher-operator helper	16.52	Sulfur-crusher helper B	16.07
Driller, pneumatic	17.50	Tripper operator-secondary crusher	16.07
Driller, pneumatic helper	16.07	Trommel helper	16.07
Dumpman	15.62	Truck driver:	
Excavator operator (unloading bridge)	20.93	Heavy haulage units	17.99
Excavator oiler:		Water trucks and trucks up to and including 10 yards	17.01
Class A	17.99	Light equipment	16.52
Class B[1/]	17.50	Truck spotter	15.62
Front-end-loader operator[3/]	16.52	Unloading bridge-hopper operator	16.07
Gantry-crane operator (Cementation launders)	18.97	Warehouse dispatcher	17.50
General maintenance:[2/]		Apprentice schedule:	
Class A	18.97	1st 6-month period	16.07
Class B[1/]	17.50	2d 6-month period	16.29
General plant operator	17.50	3d 6-month period	16.52
Helper, general:		4th 6-month period	16.77
Class A	16.52	5th 6-month period	17.01
Class B	16.07	6th 6-month period	17.50
Iron-launder helper	16.07	7th 6-month period	17.99
		8th 6-month period	18.48

1/ No employee shall be held as Maintenance Class B or Maintenance Helper Class B for a longer period than 1 year. An employee classified as Maintenance Class B or Maintenance Helper Class B will, within a period of not longer than 1 year, be advanced to Maintenance Class A or Maintenance Helper Class A, provided he is competent to perform the duties of a Class A Maintenance Man or a Class A Maintenance Helper and further provided that there is additional need in his department for Class A Maintenance or Class A Helper.

2/ Electricians, mechanics, machinists, welders, tinsmiths, carpenters, pipefitters, and painters.

3/ Whenever a front-end loader is operated in the sulfur stockpile, replacing a shovel loading sulfur, the rate for this job should be $17.99.

Remarks: Leadmen shall receive $0.125 per hour above the rate of the men led.

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First-aid instructors are Anaconda employees who teach first-aid practice and have been certified as qualified teachers by both the American Red Cross and the Bureau of Mines. Both courses are offered. First-aid classes are attended voluntarily on employee time. At present, approximately 75 employees have completed first-aid courses.

MILLING

Primary Crushing

Ore is delivered to the primary crusher in 25-ton end-dump trucks. An attendant at the crusher directs the backing of the truck and records the truck number and the load. A photoelectric cell counts the trucks as they back in to dump. Each truck driver reports the number of loads hauled, bench location, and place dumped.

Water sprays, controlled by the dump attendant, are installed at either end and immediately behind the bumper block to spray the ore stream as it drops into the dumping pocket.

The ore is dumped over an inclined grizzly made of 12-inch manganese-steel bars set with 3-inch openings between them. The oversized material feeds directly into a 54-inch Traylor gyratory crusher set at 5 inches. Undersize from the grizzly and the crushed material drop into a 150-ton-capacity receiving bin below the crusher.

The crusher is equipped with a Ducron dust-collector system, and the power for the crusher is obtained from a 400-hp. motor.

No manual feeding of the crusher is required. It is capable of crushing a 25-ton load in 45 seconds. If large rocks bridge over the crusher opening, a rock hook on an overhead crane is readily available to remove them and get the feed started again.

From the receiving bin below the crusher, a 48-inch conveyor belt (No. 1), fed by a 60-inch pan feeder, conveys the minus-5-inch rock to a storage pile (surge pile) directly over four 48-inch pan feeders. Sizes and speed of conveyor belts are shown in table 3. This storage pile has a live capacity storage of 9,000 to 11,000 tons, plus 6,000 to 7,000 tons dead storage; it is from this storage pile that ore is obtained for filling a leaching vat when the mine is not operating. The storage pile also acts as a blending pile, as the different grades of ore from the pit are well mixed by the piling and the subsequent drawing off from the bottom.

TABLE 3. - Crushing-plant conveyor belts - size and speed

No.	Width, inches	Horizontal distance, feet	Speed, feet per minute 1/
1.....	48	512	475
2.....	48	476	350
3.....	42	309	550
4.....	42	37	550
5.....	42	326	550
6.....	36	201	550
7.....	36	1,316	550
8.....	36	150	576

1/ Based on plant capacity at 1,000 tons per hour.

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Secondary Crushing

Ore from the coarse-ore storage pile is delivered to two 5- by 12-foot double-deck screens. The minus-5-inch material is delivered to a 48-inch conveyor belt No. 2 by four 48-inch pan feeders, 2 of which operate simultaneously. Alternating the use of the 4 pan feeders throughout a crushing shift minimizes the effects of segregation in the storage pile. The No. 2 conveyor belt carries the ore under a tramp-iron magnet to the screens. The top deck is a 1-1/4-inch rod screen with 1-1/4-inch-square openings. Slotted screen cloth with 7/16- by 4-1/2-inch openings is used for the bottom deck, with the slots at a right angle to the flow of ore.

The oversize (plus-1-1/4-inch) from the top deck goes to a 7-foot standard cone crusher set at 1 inch; the crushed product goes to a 42-inch conveyor belt (No. 3). The plus-7/16-inch, minus-1-1/4-inch rock bypasses the crusher and also goes to the 42-inch conveyor belt (No. 3). The minus-7/16-inch material goes to a 36-inch conveyor belt, No. 6 (fig. 13).

Conveyor belt No. 3 carries the minus-1-1/4-inch, plus-7/16-inch ore to the junction house, where it is dropped onto another 42-inch conveyor belt (No. 4) and the direction of ore flow reversed. Conveyor belt No. 4 transfers the ore to another 42-inch conveyor belt (No. 5). Conveyor belt No. 5 carries the ore to the top of the secondary crushing plant, where a motor-driven tripper drops the ore into three 300-ton steel storage bins.

The ore is fed by a 48-inch pan feeder from each 300-ton storage bin to an inverted "Y" chute, which divides the ore stream; half goes to each of 2 vibrating, double-deck screens ahead of a short head crusher.

Each screen is equipped with mesh cloth with 3/4-inch square openings on the top deck and 7/16-inch by 4-1/2-inch Tyrod cloth on the bottom deck. The screens are set at a 22.5° slope and operate countercurrent flow at speeds varying from 850 to 950 vibrations per minute; the screens are operated at different speeds to prevent setting up harmonics in the building.

The screen oversize from both decks discharges into a short-head crusher at a minimum set of 1/4 inch on choke feed. The screen undersize falls through chutes to No. 6 conveyor. The short-head-crusher product falls to No. 3 conveyor belt and is returned to the 300-ton bins above the screens. The three short-head crushers operate in parallel and in closed circuit with the vibrating screens.

The ore delivered to the leaching vats is the total undersize product of the 8 vibrating screens - 2 at the 7-foot standard crusher and 2 each at the 3 short head crushers. The material is carried by No. 6 conveyor, then No. 7, and finally by No. 8 conveyor to the leaching vats.

The No. 6 conveyor is a 36-inch conveyor belt with an S curve in it and transports the 7/16-inch ore from the crushing plant to the sampling tower. No. 7 conveyor belt is 36 inches wide, extends 1,316 feet, and conveys the ore from the sampling tower to the No. 8 conveyor of the loading bridge over the vats.

A head sample is cut automatically from the ore stream as it leaves the No. 6 conveyor. A vezin-type cutter, 12 inches long with a 1-1/2 inch slot, passes through the ore stream every 6 minutes, cutting approximately 1,600 pounds out of each vat of ore. The sample is split in 100-pound portions by a Jones riffle, giving a sample of about 400 pounds, which is put into new paper bags and delivered to the laboratory for preparation and assay.

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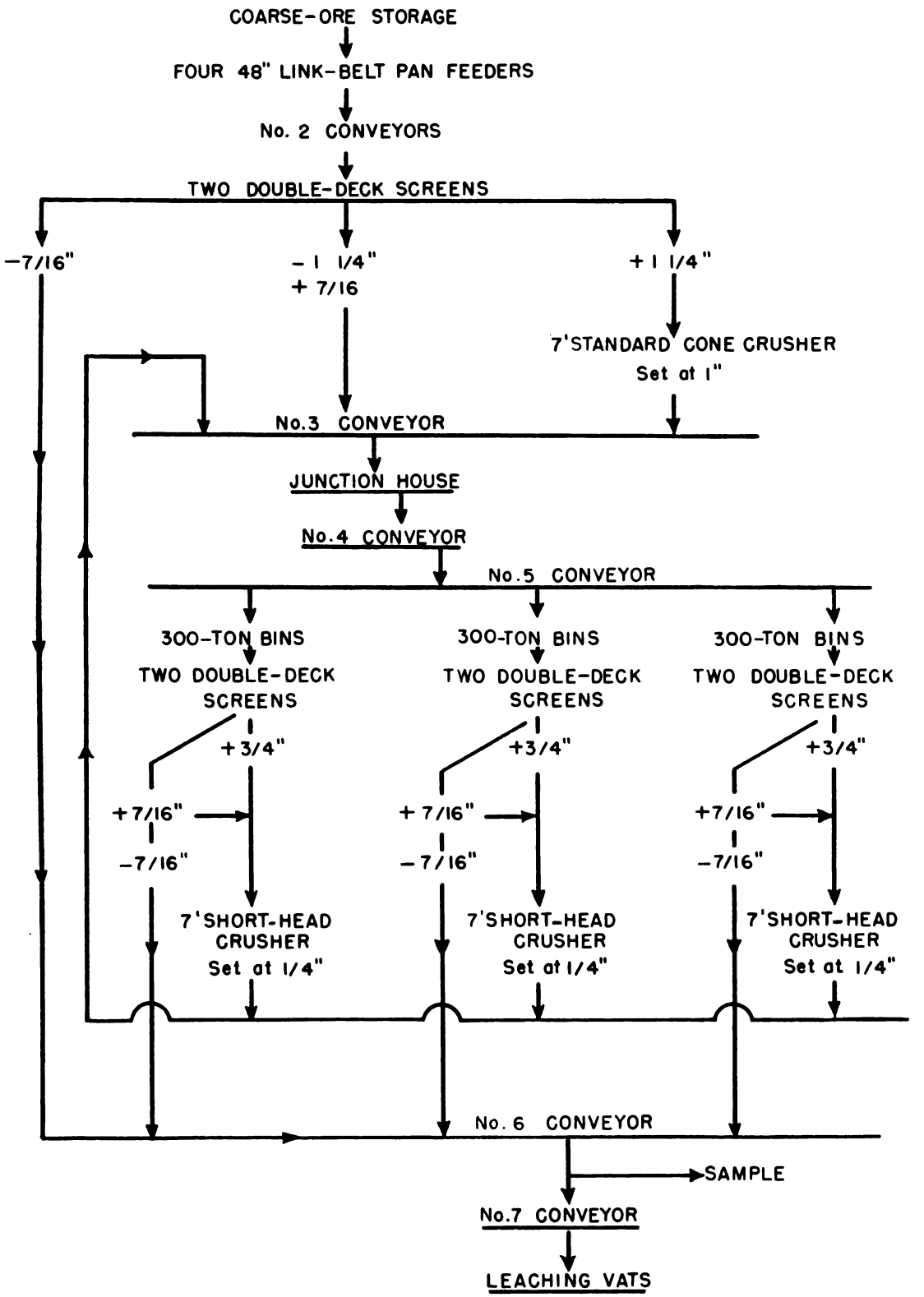

FIGURE 13. - Flowsheet of Secondary Crushing.

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Agglomeration

Preliminary experimental work on copper oxide ores was conducted many years ago under the direction of Walter Scott, plant superintendent at Inspiration Consolidated Copper Co. Additional experimental work was conducted on the Yerington ore during 1952 and 1953 under the direction of the local management and plant superintendent, A. J. Gould. This work determined the optimum size for leaching the Yerington ore to be 3/8 inch. The ore, when crushed to this size, averages 6.5 percent minus-200-mesh material, and it was decided that some method would have to be devised to prevent this fine material from forming impervious layers in the leaching vats.

The fine material was agglomerated by increasing the moisture content to 5 or 6 percent, followed by several tumbling actions in the secondary crushing plant and the conveying system. It is accomplished by wetting each layer of crushed ore as it falls onto No. 6 belt. As the ore is layered on the No. 6 belt from the chutes, each of the six layers is sprayed with water, the amount being determined by visual observation. The main criterion is that the ore stream should show no dusting as it is bedded in the leaching tanks. Each spray wets the surface of the layer of ore dropping from the chute just ahead of it, which means that the dry fines in the succeeding layer will stick to the wet, coarse particles already on the belt.

After the crushed ore leaves the secondary-crusher building, it is tumbled to obtain the proper mixing by placing an S curve in the No. 6 conveyor. The tumbling action is continued by transferring the ore from the No. 6 conveyor to the No. 7 conveyor at the sampling tower; by transferring the ore from the No. 7 conveyor to the No. 8 conveyor at the loading bridge; and by the tripper at the loading bridge, where the ore drops through a grid of wooden slats into a vat. It is bedded in the vats in 6 layers - each about 3 feet thick over only one-third of the vat area. Starting at the south wall, one-third of the floor is covered with ore 3 to 4 feet thick. The bridge returns to the south wall and beds that one-third to a point where the ore will not roll down the slope of the ore bedded on the previous pass; this is repeated until that one-third area is full. The procedure is then repeated on the middle and north third of the vat.

The agglomeration and method of bedding the vat insure that there will be no segregation in the vats, no complete blanket of fines, and therefore, no channeling of solution or complete blockage of circulation.

LEACHING

The four basic essentials of good leaching are:

1. To have an ore column through which solutions percolate freely.

2. To allow enough time for the acid-bearing solutions to penetrate the largest size material.

3. To apply solutions of adequate acid strength and temperature to dissolve the copper effectively without taking an excess of impurities into the solution.

4. To wash the leached ore free of copper-bearing solution.

In all four points a high degree of success has been obtained at Yerington. Specific operational data are summarized in table 4.

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TABLE 4. - Operational data on leaching and cementation

Acid consumption per ton of ore, 100 percent sulfuric acid.....pounds	75
Copper extraction in leaching.....percent	88 to 90
Copper recovery in cementation.....do.	99.65
Iron consumption (cementation) per pound of copper precipitates.....pounds	1.25
Copper contained in pregnant solution.....grams per liter	20 to 25
Sulfuric acid contained in pregnant solution.....do.	2 to 8
Ferric iron contained in pregnant solution.....do.	7 to 9
Strength of covering solution, sulfuric acid.....do.	20 to 35
Strength of covering solution after first and second advance H_2SO_4.....do.	40 to 50
Strength of first wash.....do.	3
Copper produced per day.....pounds	183,333
Average assay of cement copper.....percent	86.27

The 8-day cycle of bedding the ore, leaching, draining, and excavation of residue is carried out in sequence in eight 12,000-ton tanks. Each leach vat measures 120 feet by 135 feet inside measurements at the top. A 16-inch filter bottom and an allowance for freeboard at the top make the available depth of the vat 18.8 feet. Each tank holds about 12,000 tons of ore and 800,000 gallons of solution.

The tanks are built of reinforced concrete. Wall thickness at the top is 1 foot, but wall batter produces a 3-foot thickness at the bottom. To protect the concrete from the corrosive action of the sulfuric acid, mastic lining is used, consisting of 30 percent of asphalt and 70 percent of sand, reinforced with 2 layers of 6-inch, square-mesh, No. 2-gage welded wire.

The filter bottom is built with 3 layers of timber stringers and 2 layers of cocoa matting. The bottom layer is made up of 6- by 6-inch, 12- and 16-foot timbers spaced on 16-inch centers, leaving a 2-inch space between the ends, allowing the solution to drain to 3 transverse and 2 longitudinal drain channels. Three- by three-inch timbers of various lengths are placed 2 inches apart at right angles over the above. Two layers of cocoa matting are then nailed on the 3- by 3-inch timbers. Four- by six-inch timbers are then placed over the matting on 3-foot centers to protect the matting when the vats are unloaded.

One leaching vat is scheduled to be bedded and ready to start its leaching cycle at midnight. The ore in the vats varies in grade from 0.65 to 1.20 percent of copper, but the overall average for 1 month is 0.82 percent. This variation causes no particular difficulty in the leaching circuit; however, it is necessary to increase the acid strength of solution in the vat having a higher grade ore from 5 to 10 grams per liter above normal on the second day of the leaching cycle. It causes the grade of copper solution going to the iron launders to fluctuate, complicating that operation somewhat when the copper content exceeds 25 to 30 grams per liter.

The leach cycle is explained by describing the complete leaching cycle of one vat. All vats are treated similarly, with a few minor exceptions. Table 5 shows the approximate leaching schedule for a complete cycle of ore in one vat.

Leaching may be in a 96- or a 120-hour cycle; in either, it may be divided into 24-hour periods. The 96-hour schedule has 4 leaching periods, plus 32 hours washing period; the 120-hour schedule has 5 leaching periods, plus 36 to 40 hours washing period. The bedded vat (No. 4) of ore starts its leaching period at midnight. It is covered with a 20- to 25-gram-per-liter solution of sulfuric acid, which is advaned from the vat ahead (No. 3). An upward and downward percolation combination is used for covering the ore with the solution, which requires about 6 hours. Upward

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percolation from the No. 3 vat is used to force out much of the air and continues until back pressure from the hydrostatic head and bed materially slow addition of solution. The bottom feed valves are then closed, and the remainder of the covering solution is added at the top.

TABLE 5. - Approximate schedule for complete leaching cycle of ore in one vat

Period and operation	Time, hours	Acid strength, grams per liter	Remarks
Bedding (loading)	12-14		
First leaching period (22-24 hours)			
Cover	5-6		
Recirculation	3	5 or less	
Displacement	6		To copper-solution storage.
Recirculation	7-9	20-25	
Second leaching period (21 hours)			
Advance	5		Cover next tank and continue as at left.
Recirculation	3	50	
Advance	6		
Recirculation	7	40-45	
Third leaching period (21 hours)			
Advance	5		
Recirculation	3	40-45	
Advance	6		
Recirculation	7	40-45	
Fourth leaching period (21 hours)			
Advance	5		
Recirculation	3	40-45	
Advance	6		
Recirculation	7	25-30	
Wash period (32-40 hours)			
First wash	5		
Recirculation	3		
Second wash	6		
Recirculation	7		
Third wash	5		
Drain	6		To wash-solution basin.
Excavating residue	14-16		

The solution is then recirculated for 3 hours, during which time the solution collecting in the drain channels is pumped through an airlift and reenters at the top of the tank. This period of recirculation takes the solution almost to acid neutrality as the copper is dissolved. The pregnant-copper solution flows to two 286,000-gallon storage tanks. Removal of pregnant solution is continued until the acid concentration in the underflow builds up to approximately 8 grams per liter, at which time vats go on another recirculation cycle for 7 to 9 hours. The next 60 to 96 hours consists of alternating displacements and recirculation periods. During the recirculation periods 93-percent sulfuric acid is added until the desired concentration of 40 to 60 grams per liter is reached.

Three wash cycles follow leaching; the first, or advancing wash water, consists of advancing filter product from the No. 3 vat, which has just received its first wash. The second and third washes are with slurry from the Peabody scrubber in the sulfuric acid plant, which contains 3 grams per liter of sulfuric acid. Drainage of the third wash requires about 4 hours.

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One million gallons of pregnant solution per day is sent to the iron launders.

Acid consumption is 75 pounds of 100-percent sulfuric acid per ton of ore. Leach extraction of the total copper in the ore is 88 to 90 percent, or about 93 percent of the soluble copper in the ore.

Residue remaining after the leaching and washing process is dug from the vats with an 8-ton clamshell bucket and discharged into two 50-ton hoppers mounted on a traveling unloading bridge. The hoppers discharge into 25-ton end-dump trucks (these are the same trucks that are used for ore hauling). Four trucks usually are employed on the afternoon and the graveyard shifts to haul the residue to the waste dumps.

The residue sample is accumulated from a measured amount taken from every second truck loaded from the inside hopper. The sample accumulates to a total of 300 to 400 pounds for each vat, placed in paper bags, and delivered to the assay department.

CEMENTATION

The cementation plant consists of 20 parallel launders (fig. 14), each divided into 2 sections by a concrete wall butted at one end to the common (west) wall and extending to within 6 feet of the other wall. This opening allows the solution to flow from one section to the other within the launder. Each section is 10 feet wide by 58 feet long, 4 feet, 3 inches, deep at one end and 5 feet, 6 inches deep at the other. The launders are set on 1.25-percent slope, each being a little lower than the preceding, to facilitate flow from one launder to the next. A solution-distributing launder is built along the full width of the ends of the launders.

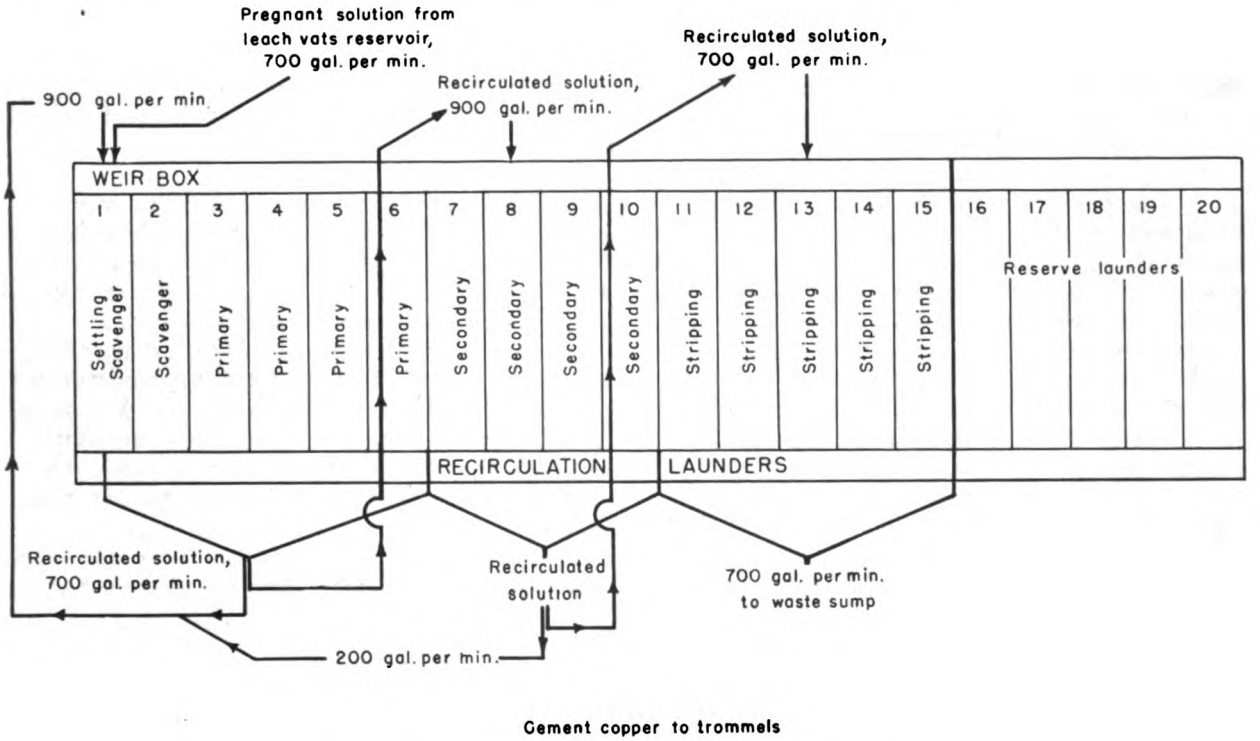

FIGURE 14. - Flowsheet of Cementation Launders.

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Solution may be fed to the cementation launders by:

1. Overflow weirs, the solution draining through any of the 3 weep gutters running the full length of each section.

2. An upward weep in both sections and draining to the next launder through a 6-inch bleed hole in the wall.

3. Upward weep in the A section and downward weep through the B section of each launder.

The launders are divided into groups: Section 1A is the settling basin for wash water and water returning from the trommels; sections 1B, 2A, and 2B are the scavenger launders; sections 3 to 6, inclusive, are the primary launders; launders 7 to 10, inclusive, are the secondary, and the launders 11 to 15 the stripping launders. The remaining launders are spares and may replace any of those normally used in the precipitation cycle.

The water returning from the trommels is caught in one section of No. 1 launder and the clear water pumped into the recirculation launder. Undigested iron scrap recovered by the trommels is placed in the other section of No. 1 and both sections of No. 2 launder. Solution fed to these scavenger launders is the same as that fed to the primary launders. The primary purpose of the scavenger launders is to consume the undigested scrap iron recovered from the other launders by the trommel. Two of the three sections are in circuit, while the other is being cleaned of precipitate.

The primary group is composed of 4 launders, 3 of which receive the solutions, while the fourth is being cleaned of precipitates and recharged with scrap iron. The secondary group consists of 4 launders, 2 of which receive recirculated solution while the other 2 are being cleaned and recharged. The stripping group consists of 5 launders, 2 launders at a time receiving the solution after it is discharged from the secondary group and the other 3 being prepared to take solution.

New pregnant solution at the rate of 700 g.p.m. (gallons per minute) is pumped from the storage reservoir to the weir box at the head of the distribution launder, where it is joined by 900 g.p.m. of recirculated solution. The solution percolates upward through the primary launders, which are three-fourths filled with new scrap iron, and overflows into the recirculation launder at the rate of 1,600 g.p.m. Seven hundred g.p.m. of the discharge from the primary launders is returned to the head of the circuit. Nine hundred g.p.m. is fed to the secondary bank of launders. The discharge from the secondary bank is directed to a sump, from which 700 g.p.m. is pumped to the stripping launders and 200 g.p.m. joins the recirculating solution going to the primary launders. The discharge from the stripping launders flows to a 30,000-gallon waste sump, from where it is pumped to the acid plant to sluice the calcines and dust residues to the evaporation area.

Table 6 gives the average assay values of solutions at the various points in the circuit.

Each primary launder is kept in the cementation circuit about 4-3/4 days, each secondary launder about 6-3/4 days, and each stripping launder about 15 days. The launder is taken out of the circuit and washed 8 to 12 hours, using a volume of 5 times as much water as cement copper, and drained. The cement copper is excavated with a gantry crane, equipped with a 6-ton clamshell bucket, and dumped into 8-ton

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hoppers, which feed two 5- by 18-foot trommels. The copper is washed through the openings in the trommel to a 36- by 36- by 7-foot deep settling basin. Discharge from the end of the trommel consists of washed scrap iron, which falls into another pit from which it is recharged to the scavenger launders.

TABLE 6. - Average assay values of solutions at the various points in the cementation circuit

	g.p.m.	Cu, gm./l.	H_2SO_4, gm./l.	Fe, gm./l.	Fe^{+++}, gm./l.
Primary and scavenger launders:					
New solution	700	20.0	5.8	7.2	5.4
Recirculated solution	900	3.5	2.4	23.6	1/.5
Total feed	1,600	10.7	3.8	16.4	2.6
Discharge	1,600	3.8	2.5	23.2	Trace
Secondary launders:					
Recirculated solution (feed)	900	3.5	2.4	23.6	.5
Discharge	900	1.0	2.1	26.4	(2)
Stripping bank:					
Feed	700	1.0	2.1	26.4	-
Discharge	700	.05	2.0	28.1	-

1/ The recirculated solution in the primary and scavenger launders is the same strength as the recirculated solution in the secondary launders.
2/ The discharge solution in the secondary launder is the same strength as the feed solution to the stripping bank.

The cement copper in the settling basin is transferred to two 30- by 30-foot drainage basins with cocoa-mat filter bottoms, then to 2 oil-fired drying hearths where the moisture content is reduced to less than 20 percent. Table 7 gives metallurgical data for September 1956 at the leaching and cementation plant. The cement copper is loaded from the drying hearths into specially designed trucks and hauled to Wabuska. The trucks return from Wabuska with purchased iron scrap.

Sampling at Wabuska

At Wabuska the cement copper is loaded into railroad cars with a Hough Payloader and shipped to the Aanconda smelter at Anaconda, Mont.

A small measured amount of precipitates are taken from each scoopful of the Hough Payloader as the car is loaded, combined into 1 sample for each car, split once on a Jones riffle to about 50 pounds, and delivered to the assay laboratory for moisture test and assay.

COST DISTRIBUTION

Information concerning the cost distribution for the various phases of mining, crushing, and precipitation of copper is shown on a percentage basis in table 8. These percentages include additional waste removal accomplished during the latter part of 1956.

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TABLE 7. - Monthly metallurgical report for June 1956

Heads:		Acid consumed.....tons	13,180
Ore charged.....(wet) tons	394,674	Acid consumed, 100 percent H_2SO_4 per ton ore bedded.....pounds	71.63
Moisture after agglomeration.....percent	6.757	Acid consumed, 100 percent H_2SO_4 per pound Cu precipitate.....do.	4.80
Ore charged.....(dry) tons	368,006		
Oxidized Cu acid-soluble.....percent	0.738	**Precipitation launders:**	
Other Cu.....do.	.105	Total solution to launders.....gallons	30,328,754
Total Cu.....do.	.843	Acid to launders.....(average) gm./l.	7.5
Ore on 4-mesh.....do.	55.4	Acid from launders.....do.	1.3
Ore on 20-mesh.....do.	82.5	Fe to launders.....do.	10.850
Ore through 200-mesh.....do.	6.1	Fe from launders.....do.	35.230
		Fe^{+++} (ferric iron) to launders.....do.	7.1
Tailings:		Fe^{+++} (ferric iron) from launders.....do.	0
Excavated.....(dry) tons	368,696	Cu to launders.....do.	21.735
Moisture.....percent	11.844	Cu from launders.....do.	0.010
Oxidized Cu acid-soluble.....do.	.061	Efficiency, Cu recovery.....percent	99.954
Other Cu.....do.	.036	Fe consumed per pound copper (estimated).....pounds	1.283
Total Cu.....do.	.097	Fe consumed per pound copper (actual)..do.	1.284
Oxidized Cu extraction.....do.	92.41	Cu precipitated.....do.	5,486,487
Other Cu extraction.....do.	60.87		
Total Cu extraction.....do.	89.17	**Acid production:**	
Dissolved Cu per ton ore.....pounds	15.98	Total operated.....days	29.02
		Acid as 100-percent H_2SO_4.....tons	13,074
Leaching circuit:		Sulfur ore burned.....do.	17,063
Total.....gallons	39,676,800	Moisture in ore.....percent	6.2
Acid.....(average) gm./l.	28.0	Sulfur in ore.....do.	30.21
Cu.....do.	11.694	Conversion efficiency SO_2 to SO_3.....do.	97.05
Fe.....do.	10.0	Conversion efficiency sulfur in ore to acid.....do.	88.50
Fe^{+++} (ferric iron).....do.	6.4		
Temperature.....(average) ° F.	92.3		
Mean atmospheric temperature.....do.	70.0		

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TABLE 8. - Cost distribution, Yerington mine, on percentage basis

	In percent of total cost			
	Power	Labor	Supplies	Total
Development: Development, contract drilling from surface	-	1.4	1.4	2.8
Mining:				
Loading (ore and waste)	0.3	3.2	3.0	6.5
Hauling (ore and waste)		5.5	5.2	10.7
Road and/or bench maintenance		.8	.5	1.3
Blast hole drilling	.1	.6	.7	1.4
Blasting and secondary drilling		.2	1.2	1.4
Water control: Wells (drilling, pumping and maintenance)	.5	.5	.2	1.2
Crushing:				
Primary crushing	.2	.6	.2	1.0
Secondary crushing and screening	.6	2.2	1.0	3.8
Leaching:				
Preparation of vats and bedding of ore	.1	.6	-	.7
Leaching (including cost of acid)	1.9	1.3	25.7	28.9
Removal of residue	.2	2.4	2.1	4.7
Cementation:				
Precipitation and recovery of precipitate		1.4	22.1	23.5
Loading and hauling precipitate		.4	.2	.6
Other:				
Engineering, assaying, supervision, and fire control		3.7	.7	4.4
Miscellaneous and general expense	.3	.9	5.9	7.1
Total	4.2	25.7	70.1	100.0

SULFURIC ACID MANUFACTURE

Mining

Sulfur ore is mined by The Anaconda Co. from the Leviathan mine, Alpine County, Calif., 58 miles west of Yerington, Nev. The ore is hauled to the FluoSolids acid plant at Weed Heights, Nev.

The ore body at the Leviathan mine is approximately 100 feet thick and averages 28 percent elemental sulfur; it is covered with a 300-foot thickness of overburden. Mining and hauling of the sulfur ore are accomplished during the summer months at the rate of 1,600 tons per day.

The stripping of the overburden is contracted to the Isbell Construction Co. of Reno. The sulfur ore is mined by Anaconda, and haulage of the ore from Leviathan to the Yerington stockpile is contracted to the Wells Cargo Trucking Co. The ore is hauled with a fleet of 15 bottom-dump, semitrailer trucks hauling 22 tons per trip.

Sulfur ore, averaging 25 to 30 percent elemental sulfur, is crushed at the mine to minus-5-inch size and stockpiled near the acid plant at Weed Heights. The minus-1/8-inch material is discarded at the mine. During the hauling season the stockpile is maintained with a bulldozer and motor grader to facilitate dumping. This equipment mixes the ore and accomplishes some drying, resulting in a uniform grade of feed with a desirable moisture content to the sulfur-ore-crushing plant.

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Crushing

Six hundred tons of sulfur ore, averaging 6 percent moisture, is hauled each day from the stockpile to a 30-ton surge bin ahead of the primary crusher (fig. 15). A 36-inch pan feeder delivers the minus-5-inch ore from the surge bin to an 18- by 36-inch jaw crusher set to crush to minus-3-inch. The minus-3-inch material is conveyed by conveyor No. 10 to a 4-foot Symons short-head crusher set at minus-1-inch. The No. 11 conveyor receives the minus-1-inch material and delivers it to four 300-ton, inverted, 60°, cone-shaped steel bins. Ore from each bin is fed by a Transporto feeder to a 5- by 10-foot Marcy rod mill, which dry-grinds the ore to 10-mesh. One rod mill for each reactor grinds 6.5 to 10 tons of ore per hour, the rate of feed being determined by the sulfur content of the ore going to each of the 4 Dorrco Fluo-Solids reactor units. The rod-mill discharge is carried from each rod mill to the top of the reactors by four parallel conveyors, from which the ore is dropped into a Stephens-Adamson star feeder for delivery into the reactors through an 8-inch pipe at the perimeter of the reactor dome.

Reactors

The reactors, when started, are preheated with propane gas to 900° F., while fluidizing air is occasionally introduced to insure an even temperature throughout the bed. When the whole bed reaches a temperature of 800° F. or above, feed to the reactor is started and the proper amount of fluidizing air applied. Oxidizing of the sulfur soon brings the temperature within the reactor to 1,100° F., where it is automatically held by the introduction of water through sprays in the dome of the reactor, and by air necessary to keep the bed fluid and to complete roasting.

Each reactor is 18 feet in diameter and 25 feet in height, and is lined with 2 layers of brick, 1 of insulating and the other of fire brick. The FluoSolids bed is kept at a depth of about 5 feet near a temperature of 1,100° F. The ore is in the reactor about 5-1/2 hours; the burned ore or calcines are removed from the bottom through Dorr Co. Fluo-seals.

Gases leaving the reactors, containing 10 to 12 percent sulfur dioxide, enter the cooling towers, where the temperature is lowered to 950° F., and then pass through 3 banks of Buell cyclones, the Peabody scrubber, and the Cottrell precipitators. The gases enter the tower at the rate of 30,000 cubic feet per minute, containing 8 percent SO_2 at 110° F.

The dusts removed from the gas by the coolers, cyclones, and the Peabody scrubber are sluiced to the evaporation area, along with the calcines from the reactors by using the spent solution from the cementation launders. The precipitate from the Cottrells is treated to recover selenium. The water from the Peabody scrubber is used as the second and third washes in the leaching vats.

Sulfuric Acid Production

The SO_2 gas enters the contact acid plant from the acid tower, going through the primary and secondary converters, where the SO_2 is converted to SO_3 with three heat exchangers in the circuit. The SO_3 gas goes through 2 heat exchangers and enters the adsorption tower containing 98 percent H_2SO_4 where it combines with the 2-percent water to form acid. The acid is diluted to 93-percent sulfuric acid and pumped to two 2,500-ton storage tanks. The acid is pumped from the storage tanks through two 250-gallon measuring tanks to the leaching vats as needed, usually about 400 tons per day.

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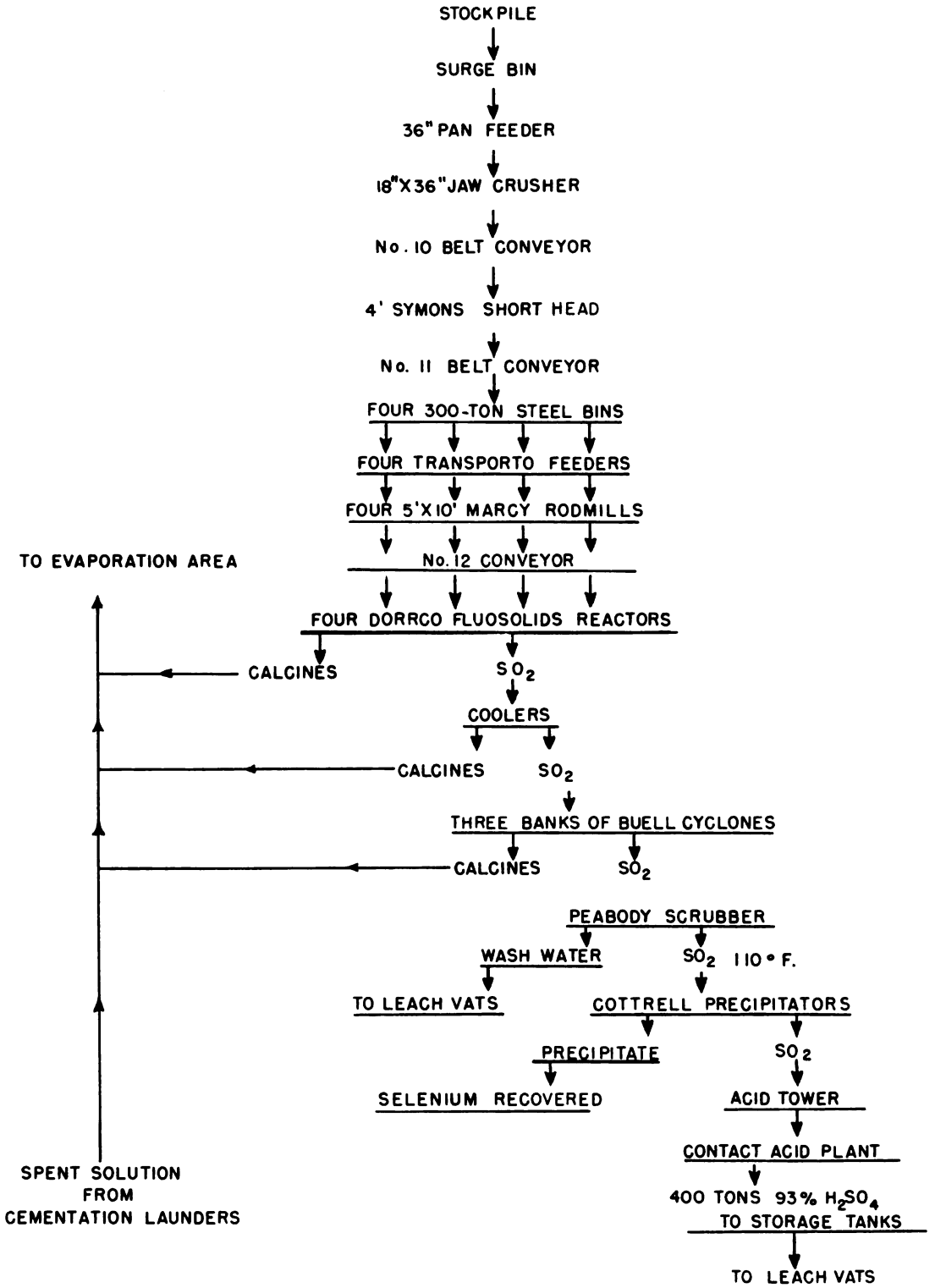

FIGURE 15. - Flowsheet of Acid Plant.

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